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Proceedings of the 2005 Coal Operators' Conference

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COAL2005
Moving Technology – Maintaining Competence
6th Australasian Coal Operators’ Conference
26 – 28 April 2005
Brisbane, Australia
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We would like to thank the following technical referees for their contribution towards enhancing the quality of papers included in this volume.

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Foreword

On behalf of the Southern Queensland and Illawarra Branches of The AusIMM, I extend a warm welcome to delegates who are attending the 6th Australasian Coal Operators’ Conference. This is the first time this conference has been held outside of NSW and the Queenslanders have welcomed this opportunity with enthusiasm.

The coal industry is under great pressure due to the recent sharp increases in world coal pricing. Our industry faces an extreme shortage of professionals and skilled labour, both of which will take many years to recover in terms of developing the required skills which come from years of experience.

In resolving the current shortage of professional people we need to address simultaneously the issues of attraction, retention and maintenance of competency. This needs to be approached on a partnership basis by the key stakeholders including the companies, the educational institutions and the profession including The AusIMM and the Mine Managers Association of Australia. COAL2005 is an example of a joint initiative which provides for the Continuing Professional Development of members of The AusIMM, ACARP and MMAA in response to the demand for advances in technology.

This year, we have successfully compiled an extensive range of topics and papers such that we are running parallel sessions. We have also been fortunate to incorporate into the program an outstanding series of papers for SPONCOM 2005. Both streams of papers have been chosen in light of the conference’s theme ‘Moving Technology – Maintaining Competence’.

All papers presented have been peer reviewed in accordance with The AusIMM’s guidelines to ensure our conference standards remain among the best regarded in the technical world. We are confident that our conference volume will serve as a valuable reference for years to come.

As a bonus, we have compiled a CD of the proceedings from all the Coal Operators’ Conferences held to date. This is available for purchase at the Conference, and afterwards from The AusIMM’s offices in Melbourne.

I wish to take this opportunity to express my gratitude to the organising committee for their hard work in providing the invaluable technical input, to the paper reviewers who have performed a difficult and professional task, and to the staff at The AusIMM office in Melbourne who have provided their extensive expertise in ensuring these conferences are held to the highest standards.

I would also like to thank The AusIMM’s Illawarra Branch, ACARP and the Mine Managers of Australia Association (MMAA) for their solid support in promoting this conference.

Finally I wish to acknowledge the generosity of the sponsors and exhibitors for their financial assistance which enables The AusIMM to present these conferences at affordable fees.

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The Effect of Resin Thickness on Bolt-Grout-Concrete Interaction in Shear

N Aziz¹, J Hossein¹ and M S N Hadi¹

ABSTRACT

Numerical modelling is extensively used in civil and mining engineering applications because of cost and risk problems associated with different experimental studies. In this study, the effect of resin thickness was evaluated in bolt-grout-concrete interaction and bending behaviour of a fully grouted bolt installed across joints in post failure region. Tests were conducted in 20 and 40 MPa concretes, and modelling simulations were made, using ANSYS version 8.1, to include both with and without different pretension loads. It was found that in all resin thickness, both the strength of the concrete and bolt pretension had major influences on the shear resistance and shear displacement of the reinforced medium. Also it was found that the strength of the surrounding concrete is more important than that of the grout thickness in both the shear resistance and shear displacement when the bolt is pretensioned.

INTRODUCTION

Rock bolts are one of the most popular systems of support in underground mining and tunnelling operations. Bolts can be installed as passive or as active pre-tensioned element. However, there is an ongoing debate on the methodology of bolt installation with regard to bolt pretensioning. When bolts installed in underground excavations or in surface mining, they are loaded both axially and laterally depending on the movement direction of reinforced unstable rock blocks.

Research on sheared surface reinforcement has been pursued, with increasing vigour in recent years, since the benefits of full encapsulation was realised and appreciated. A lot of alternative methods and experimental programs were performed in order to investigate the mechanical behaviour of fully grouted rock bolt in field of shearing such as: Dulacska, 1972; Bjurstrom, 1974; Azuar, 1977; Nitzsche and Hass, 1976; Hass, 1981; Hibino and Motojima, 1981; Dight, 1982; Spang and Egger, 1990; Ferrero, 1995; Pellet and Boulon, 1998; Kharchafi, Grasselli and Egger, 1998; Grasselli, 2004; Aziz, Jalalifar and Hadi, 2004 and Jalalifar, Aziz and Hadi, 2004.

The effect of resin thickness is one of most important factors to transfer the maximum load from bolt to the rock. When a bolt is loaded axially by pushing/pulling through the resin thickness, thicker resin layers show minimum load transfer. Several authors conducted this research in the past (Fabjanczyk, Hurt and Hindmarsh, 1998; Aziz and Webb, 2003; Aziz, 2004). However, to understand the effect of resin thickness while bolt is laterally loaded under shearing effect, laboratory tests and numerical simulations were conducted. Double shear laboratory tests were carried out only in 27 mm hole diameter and 22 mm bolt diameter. Afterwards an extensive numerical modelling with different resin thickness, material strength and bolt pretensioning was undertaken. This paper highlights the effective role that the resin thickness in different rock strengths and pretensions could play in rock reinforcement.

EXPERIMENTAL STUDY

Figure 1 shows the general set up of the double shear box unit in a testing machine. Figure 2 shows the sketch of the quarter model of double shear block, the axial section of the assembled reinforced block with various induced forces and compression and tensile zones along the bolt. Double-jointed concrete blocks were cast for each double shearing test. Two different strengths concrete blocks, at 20 MPa and 40 MPa, were cast to simulate two different strength rocks.

The concrete/bolt assembly was then mounted in a steel frame shear box specifically fabricated for this purpose. A base platform that fitted into the bottom ram of the Instron Universal Testing Machine, capacity 500 kN, was used to hold the shear

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box. A predetermined tensile load was applied to the bolt prior to shear loading. This acted as a compressive/confining pressure to simulate different forces on the joints within the concrete. The three nominated tension forces used were 20 kN, 50 kN and 80 kN. Axial tensioning of the bolt was accomplished by tightening simultaneously the nuts on both ends of the bolt. The applied axial loads were monitored by two hollow load cells mounted on the bolt on either side of the block. Figure 3 and 4 show the load deflection trend for bolt type T1 in 20 and 40 MPa concretes respectively and in different pretensions installed in 27 mm hole diameter. Figure 5 shows the axial load developed along the bolt type T1 during shearing. Different graphs depict different initial pretension conditions. The rate of axial load increase is on the rise once the elastic range is exceeded. Back sloping in the graph of high initial pretension bolt load cast in weak concrete medium of 20 MPa is caused by the end crushing of the concrete leading to axial displacement and loss of pretension load. The shear load versus shear displacement of bolt type T1 in different pretensioning conditions is shown in Figure 6. It shows the relationship between the axial loads developed and shear displacement in bolt type T1, installed in 40 MPa concrete. From the graph it can be seen that the axially induced load on the bolt at low level of pretensioning, is higher than that of high level of pretensioning. The possible reason is attributed to lower confinement resistance, causing the bolt to deform much more readily, with an increase in axial load of the bolt as the bolt is clamped at both ends.

**CONFIRMATION OF THE MODEL**

Initially, the numerical model was calibrated by the laboratory test and after confirmation of the shear load-deflection curves, numerical simulations were carried out in 20 and 40, MPa concrete. The objectives were to examine the strains and stresses developed with different resin thickness and various pretension loads of 20, 50, and 80 kN. Twenty-four numerical models were created, to describe different resin thickness in different pretensions and rock strength. The confirmation of numerical simulations with laboratory tests is shown in Figure 8. It can be seen that the numerical simulations were found to be in close agreement with the experimental results.

**3D NUMERICAL SIMULATION**

3D FEM of the reinforced structure subjected to the shear loading was used to examine the behaviour of bolted rock joints, such as: strength of material, pretensioning and resin thickness in relation with the experimental results. Parameters considered were, the three governing materials (steel, grout, and rock) with two interfaces (bolt-grout and grout-rocks). Using ANSYS (Version 8.1), it was possible to simulate specifically the...
elasto-plastic materials and contact interfaces behaviours. The stress-strain relationship of steel was assumed as bilinear kinematics hardening model and the modulus of elasticity of strain hardening was accounted as one-hundredth of the original value. The yield strength of the steel of 600 MPa was obtained from the laboratory tests.

Numerical modelling was carried out in different thickness of resin, 1.5, 2.5 and 5 mm, in borehole diameters of 25, 27 and 32 mm respectively.

Figures 9 and 10 show the created gap among interfaces in thin and thick resin layer. From the figures it can be seen that with increasing the shear load, grout is separated from the bolt and concrete in the tension zones and is compressed at compression zone. The separation gap occurred in all resin thickness. The gap in thin resin layer is more extensive than the thick resin layer and also changes in stresses, strains and displacement along the bolt and surrounding materials. Because of extensive output results from numerical simulations, only a few figures are provided in this paper. Figure 11 shows the value of induced strain along the bolt axis around the grout 1.5 mm thick. Compared to thick resin the level of strain, in both tension and compression zones, is higher and the resin is fragmented with low shear load.

Figure 12 shows the plastic strain along the thick resin layer in 20 MPa concrete. The value of strain in the vicinity of the shear joint, through the resin, is high and causes complete damage in resin. Figure 13 displays the value of induced strain along the bolt axis through the resin in 40 MPa concrete with 80 kN pretension. Figure 14 shows, there is a dramatic increase in strain change in the resin layer in the vicinity of shear joint. The value of strain, at both the axial and lateral directions in the vicinity of shear joint plane, was above the elastic yield point of the resin, which is a clear indication that the strength of resin is exceeded.
Figure 15 shows the trend of induced stress and concrete deflection along the bolt axis through the concrete block. It shows that the stresses around the concrete block edges in the vicinity of shear joint are high, thus inducing longitudinal fractures in the concrete blocks (Figure 16). This was also observed at the experimental results, and was common to all resin thickness and concrete strengths. The rate of strain changes along the bolt, with thick resin thickness in 40 MPa concrete and without pretension, is shown in Figure 17. From the numerical simulation it was found that the outer layer of the bolt was yielded, however, the middle part of the bolt cross section remained in the elastic state in different concrete strength.

Figure 18 shows the yield strain contours along the bolt axis in 40 MPa concrete with 2.5 mm surrounding resin thickness and 80 kN pretensioning. From the numerical simulation it was found that with increasing bolt pretension, the area of tensile strain expanded and distributed to the middle of the bolt. Also with increasing shearing load, the surrounding material, concrete or grout applies reaction acting on the bolt length, which is progressively increased until the bolt yield. In numerical simulation, surface-to-surface contact elements of 174 and 170 were defined for contact and target interface elements respectively.
Figure 19 shows the contact pressure contours between the bolt and grout in 40 MPa concrete. When the resultant bolt bent gap is increased the contact is separated and the contact pressure is removed. At the compression zones in the vicinity of shear joint, reaction stress will result. Figure 20 shows the trend of contact pressure changes along the interface in 20 MPa concrete, which was high in the vicinity of the shear joint.

THE EFFECT OF RESIN THICKNESS ON INDUCED STRESSES

The value of induced stresses in bolt was evaluated in different resin thickness. The behaviour of the concrete and grout were assumed as an isotropic linear material and the behaviour of steel bolt was assumed non-linear hardening. The effect of various concrete, grout and bolt modulus of elasticity in different resin thickness was investigated using the numerical simulations. Shear stress trend as a function of concrete modulus in different resin thickness is shown in Figure 21. The shear stress along the bolt in thick resin layer is lower than in thin resin layer. This trend was reduced with increasing concrete modulus. Figure 22 displays induced tensile stress versus grout modulus in soft concrete and in different resin thickness. The induced tensile stress along the bolt was decreased with increasing resin thickness and grout modulus.

Figure 23 shows the effect of concrete modulus on shear displacement in different resin thickness. Concrete strength has great effect on shear displacement in all resin thickness. However, not observed significant change in shear displacement in high strength concrete. The value of shear displacement in thin resin layer is higher than thick layer. As Figure 24 shows the influence of grout modulus is more effective than the concrete modulus in shear displacement for the variety of resin thickness.

RESULTS AND CONCLUSION

This paper demonstrated that the resin thickness plays a prominent role in bolt shearing across joints and bedding planes, however its roles in shear is less significant in comparison to the conventional axial loading and load transfer characteristics. Thus, what is important and influencing the bolt shear is the strength of resin in relation to the medium strength.

The following are some of the conclusions drawn from model simulations:

1. Von Mises strain in bolt roughly was constant in 1.5 and 2.5 mm resin thickness; however, a little reduction was observed in 5 mm thickness;
2. tensile and compression strains were slightly reduced with increasing resin thickness
3. shear displacement was reduced with increasing resin thickness;

THE EFFECT OF RESIN THICKNESS ON BOLT-GROUT-CONCRETE INTERACTION IN SHEAR

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4. with increasing resin thickness the plastic strains perpendicular to the bolt axis inside the grout were reduced;
5. compression and tensile strain along the bolt axis in concrete interface were reduced slightly with increasing resin thickness; and
6. it was also concluded that the strength of the surrounding concrete is more important than that of the grout thickness in shear resistance and shear displacement while bolt is loaded laterally.

![Graph of Shear stress vs. Concrete modulus of elasticity](image1)

**Fig 21** - Induced shear stress versus concrete modulus of elasticity in different resin thickness.

![Graph of Tensile stress vs. Grout modulus of elasticity](image2)

**Fig 22** - Induced tensile stress versus grout modulus of elasticity in soft concrete.

![Graph of Shear displacement vs. Concrete modulus](image3)

**Fig 23** - Shear displacement versus concrete modulus in different resin thickness.
REFERENCES


Azuar, J J, 1977. Stabilization de massifs rocheux fissures par barres d’acier scellees, Rapport de recherché No 73, Laboratoire Central des ponts Chaussees, France.


FIG 24 - Shear displacement versus grout modulus of elasticity in different resin thickness.
Application of Computer Modelling in the Understanding of Caving and Induced Hydraulic Conductivity About Longwall Panels

W J Gale

ABSTRACT

Computer modelling is being used to simulate rock fracture, caving and stress redistribution about longwall panels with increasing confidence. The models are being assessed against field monitoring and have significantly increased the understanding of caving mechanics within the overburden.

This paper discusses the modelling approach and provides some examples of its application to overburden damage and induced hydraulic conductivity. Computer models used in this study simulate the fracture process in the geological units throughout the overburden. Analysis of the mining induced fracture patterns and in situ joint patterns allows an estimation of the hydraulic conductivity within the overburden. The cubic flow relationship has been used in the examples presented.

INTRODUCTION

Computer modelling is being used to simulate rock fracture, caving and stress redistribution about longwall panels with increasing confidence (Gale, Mark and Chen, 2004; Gale, 1998, 2004). The models are being assessed against field monitoring and have significantly increased the understanding of caving mechanics within the overburden.

In general, the models have been intended to assess longwall caving issues, however their application extends to ground subsidence, overburden fracture mode and mining induced hydraulic conductivity of the ground adjacent to mining operations.

The aim of this paper is to discuss the modelling approach and some examples of its application to overburden damage and induced hydraulic conductivity. Analysis of the mining induced fracture patterns and in situ joint patterns allows an estimation of the hydraulic conductivity within the overburden. The method also provides an estimation of the conductivity effects at seam level which will impact on the effectiveness of seals placed in roadways subject to abutment related deformation and fracture.

Two examples are presented to provide an overview of the approach and to highlight findings of a current ACARP project undertaking research into induced hydraulic conductivity. The first example is that of the induced conductivity created by longwall extraction and the second is the conductivity issues for seals at the seam level.

COMPUTER MODELLING APPROACH

SCT Operations has been developing the capability to undertake computer simulations of strata caving and the interaction of longwall supports within a site-specific geological setting. This capability has been developed from in-house R&D and from collaboration with CSIRO within three interrelated ACARP Projects researching longwall geomechanics.

The model is two-dimensional and represents a longitudinal slice along the central zone of the longwall panel. The code used in the model is FLAC and uses a coupled rock failure and fluid flow system to simulate the behaviour of the strata and fluid pressure/flow effects. Rock failure is based on Mohr-Coulomb criteria relevant to the confining conditions within the ground.

Computer models are developed on the basis of detailed geotechnical testing of pre and post strata failure properties. Detailed models of the geology are necessary to obtain a satisfactory simulation of the rock failure mechanics. The model simulates rock fracture and stores the orientation of the fractures. Shear fracture, tension fracture of the rock, bedding plane shear and tension fracture of bedding is determined in the simulation. The stability of pre-existing jointing, faults or cleat is also addressed in the simulations where appropriate.

The model simulates the mining process by progressively excavating approximately 1 m shears, allowing caving and then excavating the next shear and advancing the face supports. Ground movement, rock fracture zones, water pressure, longwall support load/convergence and abutment stress distributions are determined and recorded for each ‘shear’ as the longwall retreats. Ground displacements, rock fracture and stress redistributions can be assessed within various rock units and geometries about the extraction panel.

HYDRAULIC CONDUCTIVITY OF IN SITU STRATA

The hydraulic conductivity of in situ strata is a combination of:

1. The conductivity of the rock (grain) fabric of each rock unit.
2. Joint and fracture planes which cut through the rock strata units. These planes include bedding planes mobilised in previous tectonic movements.

Typically, the conductivity of the rock fabric is low and the greatest potential for flow through the rock mass is via the inherent fracture networks within the strata sections.

A range of typical conductivities measured for rock fabrics and that of joints/fractures is presented in Figure 1. This data is a compilation of borehole tests and rock core fabric tests. The results indicate that the overburden conductivity is generally controlled by the fracture patterns within the ground.

The conductivity of fractures reduces with confining pressure which closes the aperture through which fluid may flow. Laboratory testing of rock fractures has demonstrated a rapid reduction of conductivity with confining stress across the fracture plane. The actual relationship will vary somewhat depending on the fracture surface geometry and the material. The conductivity of the near surface region (<100 m) is typically high and then rapidly reduces as confining pressures reduce the aperture to is ‘residual value’.

A relationship based on laboratory and in situ testing of targeted fractures is presented in the Figure 1 for a single plane having a spacing of 1 and 5 m. This relationship tends to fit the data reasonably well and examination of roadway cuttings and highwall exposures confirms the likelihood of fracture spacing within this range. In this study, this general relationship has been used to characterise the mass conductivity in terms of fluid flow potential through the ground.

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However, it is also noted that structural zones have been intersected which can have larger frequency or greater residual apertures than the ‘normal’ joint/bedding plane surfaces. Some examples exist in the data, however it should be anticipated that the actual conductivity of such features might well be variable and locally high.

HYDRAULIC CONDUCTIVITY OF INDUCED FRACTURES

The hydraulic conductivity (water flow) of a fracture can be estimated on the basis of:

$$K = t^{1.5} \times 10^6 \text{ m/s}$$

where:

- $t$ = hydraulic aperture of the fracture.

The hydraulic aperture is generally related to the actual fracture dilation with modification due to surface roughness. The effect of surface roughness needs to be assessed for flow calculations however its effect tends to reduce when fracture dilation exceeds approximately 1 mm.

An estimate of the horizontal and vertical conductivity within the strata about longwall panels can be obtained within the large scale models of caving. To obtain this, the dilation and hydraulic aperture of the fractured strata and bedding planes is estimated and the conductivity derived.

In this manner the conductivity distribution above the panels and adjacent to panels can be estimated. The impact on aquifers and water bodies can then be assessed on the basis of the mining induced fracture networks created. The effect of in situ fracture conductivity is included in such analyses. The aim of this approach is to provide a better understanding of the fracture distributions and their impact on conductivity within the strata surrounding the longwall panel.

EXAMPLES OF SIMULATIONS

The approach is to develop a model of the strata and then excavate the panel progressively. An example showing the geology and the resultant fracture mode within the overburden for one excavation geometry is presented in Figure 2. Overburden is 190 m. The section is composed of Permian sediments and a 50 - 60 m section of Tertiary sands, clay and basalt.

The results indicate that fracture through the overburden is extensive and affects the total section to the surface. Surface subsidence is approximately 66 per cent of seam extraction. The subsidence profile obtained is presented in Figure 3 compared to actual monitored data of panels. The results are comparable and indicate that the fracture patterns created in the overburden simulate the actual caving and subsidence characteristics of the section in a realistic manner.

The horizontal conductivity and conductivity as determined from bedding plane dilation (aperture) is presented in Figure 4. The results indicate very high, horizontal conductivity localised along bedding planes throughout the overburden section. Considering the large number of horizontal flow pathways, the vertical fracture conductivity will control the overall connectivity and potential for inflow.
The overall connectivity of the overburden has been investigated by averaging the vertical conductivity in metre thick sections across the extracted panel and creating a vertical profile. In this way an overview of the potential vertical connectivity between layers can be obtained. An example profile for this supercritical model is presented in Figure 5 and indicates:

1. Variable but generally high conductivity created in the geological units within the overburden.
2. Localised zones of low induced conductivity. Jointing in these zones is likely to impact on the pathway.
3. Essentially open flow within the immediate caved zone.

The variable conductivity within the overburden is associated with localised fracture networks created within each layer. Flow will occur through a network of vertical and horizontal pathways created within the various layers of the overburden.

The conductivity within the overburden is consistent with more generalised estimates often applied to subsided panels, however the models have the ability to relate the conductivity to the fracture networks created in the geological units within the overburden.

The geological section as characterised by UCS of the material is presented in this figure together with the fractures developed as a result of mining. The roof and rib dilation is presented in terms of an extensometer plot in Figure 7. The mining induced conductivity is presented in Figure 8. The results show that at this stage the effective conductivity of the immediate area about the roadway is high and equivalent to that of sands and gravel.
The effect of increased vertical abutment loading about the roadway was examined with the addition of a low to moderate strength seal. The seal has a stiffness of approximately 10 MPa which relates to a structurally soft or foaming type seal. The seal properties are hypothetical to demonstrate the issues rather than represent any site.

As the abutment load is increased about the roadway, roadway deformation increases and the dilation extends deeper into the strata. The roadway deformation and dilation relative to an extensometer reading is presented in Figures 6 and 7 for the addition of 5 MPa. The conductivity about the roadway is presented in Figure 8 at these load stages. The stress developed in the seal during loading was typically less than 0.5 MPa (50 t/m²). The results indicate that whilst the seal itself may remain intact, there is significant potential for bypass flow within the coal and strata surrounding the seal. Even if the ground surrounding the roadway were injected prior to abutment loading, there would still be significant potential for bypass leakage, resulting from the additional dilation and deformation about the roadway.

The resultant conductivity about the roadway can be reduced by a stiffer seal material, which confines the ground and restricts dilation. The design for seals, which are required to restrict bypass leakage as opposed to dynamic pressure events, requires detailed design and monitoring of the system. Further work is required to design such systems in terms of the materials, geometry and surface preparation.

**DISCUSSION AND CONCLUSIONS**

Stress redistribution about longwall panels, caving and subsidence movements create a fracture network of bedding, shear and tension fracture planes which combine with the in situ joints and structures. The mining induced fracture network can extend outside the mined panel. Horizontal conductivity can be
significantly enhanced along bedding planes well outside the panel. Also, bedding planes can be mobilised within the near surface overburden as a result of large-scale stress redistributions within the overburden rather than due to induced subsidence movements.

The main control on the connectivity within the fracture networks is typically the vertical (subvertical) fractures connecting horizontal bedding.

A particular application of the modelling is to provide additional information on the fracture networks created about mining panels. This information can be used as part of the mine design process to evaluate the potential impacts of mine extraction geometries on aquifers and surface features.

This approach has been applied to other sites and provides results, which are consistent with the monitored behaviour of aquifers and inflows. The impact of such networks on inflow and aquifer integrity is related in part to the recharge characteristics of the aquifier relative to the outflow into the fracture network. This will vary depending on the nature of each site; however, the modelling has the ability to provide a good understanding of fracture networks created and an estimation of the enhanced conductivity within the overburden created by mining.

This type of analysis applied to flow about seals in roadways indicates the potential for ‘time dependent’ bypass within the strata and coal ribsides. The design requirements for dynamic events such as explosions are different to those referred to as bypass leakage as discussed in this paper. Design of long-term fluid seals requires further work and monitoring studies.

Research in this project is still continuing over the next six months. Ongoing work is continuing to improve the definition of fracture connectivity within the overburden.

REFERENCES


The Value of Early Geotechnical Assessment in Mine Planning

C Hanson1, D Thomas2 and B Gallagher3

ABSTRACT

Valuable data for geotechnical interpretation and integration into effective Australian underground mine planning may often be available, yet is not always fully appreciated or utilised, particularly in the early stages of mine planning or in due diligence studies. There may be considerable benefits associated with early prioritisation of geotechnical evaluation and impact on mine planning.

Unidentified, misinterpreted, or ill-defined adverse geological and related geotechnical resource characteristics can pose significant business risk to underground coal projects and operations. Preliminary resource definition in the early conceptual mine planning stages attributes significant focus (entirely warranted) on resource quality and structural geology constraints. Yet detailed geotechnical data analysis and interpretation, which may have a substantial down stream impact and sensitivity with respect to future mine planning strategies, at times is given lower priority, or scoped and resourced in the later stages of a bankable feasibility study. Through extensive mine planning experience and observation of downstream process impacts, it has been found there is often data available for geotechnical analysis that does not readily stand out or is not adequately understood or utilised, available at the early (conceptual) stages of mine planning. Part of the issue may be that exploration geologists are not necessarily experienced geotechnical engineers and do not necessarily recognise or understand all important parameters. Subject to appropriate application of experienced professionals data can be manipulated to provide key geotechnical hazard assessment at minimal cost, and provide a framework for understanding and optimising the mine planning process.

Although there is no single prescribed strategy for resource evaluation from a geotechnical perspective, potential business risks and mitigation approaches can and should be adopted at the conceptual mine planning stage. There has been a recent focus in the metals industry to provide a reporting framework for geotechnical classification of mining projects. This paper outlines the strategies and gives examples of key analyses adopted in mine planning and discusses the relative merits of adopting a reporting framework as a tool for geotechnical classification in mine planning.

INTRODUCTION

A well known, but not necessarily implemented, fact is that geotechnical assessment forms a key driver in project viability. Primary consideration should be given to the likely mine planning implications arising from geotechnical interpretation. Significant expenditure is often attributed to the acquisition of exploration data, yet at times there appears an imbalance between resources attributed to data acquisition, processing and presentation, compared with that dedicated to comprehensive interpretation and risk assessment of relevant geotechnical data and subsequent integration into mine planning processes. There is almost always relevant geotechnical detail that can be manipulated from any form of geological exploration, that should be appropriately assessed in the conceptual mine planning process onwards.

This paper outlines experience with respect to geotechnical assessment in the context of mine planning and balanced against other key drivers. It is non-specific with respect to case histories, but rather, examines generically the experiences gained through numerous sources including:

- practical operational mining experience;
- due diligence studies, in particular auditing resource and reserves and assessment of attributed valuation and risk assessment;
- designing, costing and project managing exploration programs;
- analysing and interpreting data from exploration, in particular with respect to geological interpretation and associated geotechnical analysis at all stages of mine planning;
- completion of geotechnical evaluation at concept, pre-feasibility and feasibility study levels for coal projects.

A discussion outlining specific forms of geotechnical data that can be interpreted to add significant value at the early stages of mine planning is outlined. In mine planning, it is desirable to establish an appropriate level of geotechnical risk assessment balanced against other key drivers at each stage of the mine planning process. In conclusion, the relative merits of a reporting framework for geotechnical classification of coal mining projects are debated.

THE MINE PLANNING PROCESS

Stages of mine planning

The major stages of mining project development are set out below in Figure 1. At the end of each stage, a business case is generally made to justify progression to the following stage. A subsequent increase in exploration, data compilation, analysis and interpretation and mine planning input is required as the project development process unfolds, with an associated increase in committed human and physical resources and total cost.

At each stage in the planning process, the level of certainty with respect to project value and confidence in the specific resource and reserve characteristics increases. The prime consideration is project value and ability to achieve the projected production levels, operating cost and sales price. Geotechnical aspects affect two of these three primary determinations. Key measures of project value include:

1. **Fair market** value of each project under consideration, determined by current market conditions and price.
2. The **intrinsic** value of each project under consideration, determined by current worth and potential future earning power. Intrinsic value can be assessed at a conceptual stage using appropriate Valmin Code guidelines, however, detailed intrinsic valuation is usually estimated from pre-feasibility onwards, where net present value can be attributed over a given timeframe with discounted cash flows.
3. The **strategic** value, usually reflecting a higher value attributed due to such factors as geopolitical advantages, economies of scale or reducing competition. Strategic value may also be in the context of brownfields expansions that may reduce overall unit cost of total production output and/or make exploitation of nearby deposits more attractive or more competitive to the company than its peers.
Typical project ranges with respect to the accuracy of project valuation during each of the mine planning stages are illustrated in Table 1.

*Conceptual* mine planning studies are typically based on a level of established exploration data, historical information and inferences from regional and benchmarked experience. From the perspective of project viability this level of analysis generically represents a broad-brush assessment of possible viability, considering as wide a range of alternative scenarios and options as necessary. None the less, a business case must be made to proceed to pre-feasibility, which upon approval often requires substantial commitment of expenditure to advance the project through pre-feasibility.

When assessing either a single project or considering a portfolio comprising a number of potential projects with a strategy to narrow the field for further development, there is considerable justification in utilising all available data sources and committing to comprehensive use of all valid data and key screening criteria at this time.

This is a fundamental requirement for:
- minimising costs and resources otherwise dedicated to projects or resource areas that may not ultimately be viable;
- presenting a balanced and authentic assessment of project potential such that viable projects are not overlooked at the outset, particularly with respect to previous resources where preconceptions may exist.

An analysis using appropriate valuation tools on various scheduled mine plan options is justified at this stage of the project, and is either presented as a case for proceeding with project development or otherwise discarding. When assessing larger project portfolios, a matrix incorporating value and other

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**TABLE 1**

<table>
<thead>
<tr>
<th>Type of estimate</th>
<th>Conceptual</th>
<th>Pre-feasibility</th>
<th>Full feasibility</th>
<th>Definitive</th>
</tr>
</thead>
<tbody>
<tr>
<td>Purpose</td>
<td>Indicative business case for JV</td>
<td>Establish project scope and criteria</td>
<td>JV approval</td>
<td>Project cost control</td>
</tr>
<tr>
<td>Resource status</td>
<td>Inferred to indicated</td>
<td>Indicated to measured</td>
<td>First ten years measured</td>
<td>First ten years measured</td>
</tr>
<tr>
<td>Reserve status</td>
<td>Possible to probable</td>
<td>Probable</td>
<td>First ten years proven</td>
<td>First ten years proven</td>
</tr>
<tr>
<td>Possible range of costs around central estimate</td>
<td>30%</td>
<td>20%</td>
<td>15%</td>
<td>5%</td>
</tr>
<tr>
<td>% of design effort required to produce estimate</td>
<td>0.5% - 5%</td>
<td>5% - 30%</td>
<td>30% - 45%</td>
<td>45% - 65%</td>
</tr>
<tr>
<td>Normal estimating method</td>
<td>Scaled historical data</td>
<td>Factored budget quotes</td>
<td>Engineering estimates, firm quotes</td>
<td>Engineering estimates, full take-offs</td>
</tr>
</tbody>
</table>
strategic factors may be compiled and allow for ranking and comparison across a range of projects. Quantifying, qualifying, and benchmarking project geological and geotechnical risk should be conducted at this stage. If the business case is verified, additional resources are committed to develop a project to a pre-feasibility level of assessment. A pre-feasibility level of study allows for detailed comparison of key mine planning and strategic alternatives and usually facilitates confirmation of one or two of the most attractive options presented at concept level. Cost estimates and economics should be sufficiently accurate to select options and justify expenditure to bring the project to bankable feasibility.

A feasibility study is used to secure a commitment to finance. It presents a summary of the risks and mitigation strategies allowing a company or bank to risk weight lending rates. Cost estimates and economics should be sufficiently reliable and robust for decision on project approval to be made. Project valuation accuracy should be targeted at ten to 15 per cent at this stage. Often bankable feasibility study (BFS) mine plans become set in stone. Operations personnel may use limited initiative to revise or review, particularly if not privy to or informed of the key drivers leading to the derivation of the plan. If these key drivers change, then the BFS layout, schedule, and economics should be reviewed, and if warranted, revised.

The mine planning team

In a typical mine planning process, resources are assessed based on (minimum) industry guidelines. Such guidelines include:

- Australian Standard for Metallurgical Coal Projects,
- The Australasian Institute of Mining and Metallurgy Monograph 12, and
- internal company or corporate advice or structured guidelines.

Mine planning options are formulated, and productivities and costs are assigned within an economic model and scheduled to arrive at an estimated value. This may resemble more a comparative fair market value at the conceptual stage, and an NPV through discounted cash flow/rate of return over a fixed period from pre-feasibility onwards. Care should be taken as there will often be a tendency to overestimate value at this stage, unknown conditions may present a lower hurdle rate for screening. The typical involvement of relevant parties in this process is as follows:

Qualified geologists assess the resource quality, seam characteristics and structure, provide a resource status classification and, in combination with others, devise and manage exploration programs to reach required resource status classification.

Qualified mining engineers assess reserves based primarily on geological constraints provided, usually by way of a plan from geologists. Underlying geotechnical concepts are factored in, often based on a broad assessment of regional stress data and anticipated ground conditions, from the information provided by the geologists. In general, mining engineers are responsible for generating mine planning options and economic models from which reserves are generated and classified based on the assessed recoverable (economic) resource.

Business analysts and coal quality experts traditionally have a role in assessing key economic assumptions and sensitivities flowing forward, usually in the form of market placement and exchange rate or price fluctuations.

Marketers and corporate personnel who may identify a market niche and gain commitment from buyers.

As with consideration of mine planning components and parameters, a holistic approach should be used with individual parties working together as a team, rather than in isolation in defined roles on projects, as critical for delivery of an impartial and comprehensive mine planning process.

The team of professionals dedicated to resource and reserve assessment and project valuation at progressive stages of the mine planning process will clearly depend on the nature and characteristics of the project being assessed. Consistent with the mine planning approach as previously outlined, be it open cut or underground mining assessment potential, the most important point at which comprehensive analysis and risk assessment by mining professionals with appropriate relevant experience from available data is warranted is, arguably, at the conceptual stage. This is consistent with a philosophy of presenting a balanced (and in the case of multiple projects fair comparison) of project potential.

Dedicating comprehensive expertise at this stage will assist in minimising the expenditure committed to projects, which are not ultimately viable, and reduce the potential for ill-considered relinquishment of potential projects. One of the fundamental areas to minimise downstream mining risk that should be most comprehensively assessed at the concept stage, is that associated with analysis of structural geological, geotechnical and hydrological/hydrogeological parameters.

Opportunity and constraints with respect to resource coal quality, structure (faults), and resource recovery are always (rightly) key drivers in determination of project viability and mine layout. However, other geotechnical parameters may often be overlooked at the concept stage. The first mine layout option(s) is extremely important, as it forms the blue print for each successive stage of project development. Once committed to paper, it can be difficult to change, particularly if the change results in reduced resource recovery.

In keeping with the above argument, there are major benefits in utilising and integrating a team of experienced professionals in concept mine planning studies who have a broad range of exposure and skills in:

- practical geological/geotechnical open cut or underground operational mining and exploration experience;
- conceptual through to bankable feasibility level mine planning studies and due diligence studies for an extensive range of resources and clients; and
- economic evaluation and project financing.

The major benefits in applying appropriate expertise and strategy at concept level relate to:

- providing capacity (through experience base) for formulation of hazard plans, risk ranking, and risk assessment from a comprehensive review of all available data, such that critical issues and strategies are developed and integrated into the mine plan process;
- targeting future exploration and scoping feasibility studies to ensure that critical issues are addressed in appropriate depth and in a timely fashion with respect to landmark requirements in project development; and
- evaluating and comparing mine planning options and sequences incorporating assessed geotechnical risk parameters against other key drivers such as optimising resource extraction, resource quality and economic return.

Assessment of parameters

With a suitably selected team, preliminary assumptions and measured risks relating to the parameters assessed from available data can then be developed. The key in achieving a balanced assessment of parameters is to integrate the major components under the same analysis, rather than treat each in isolation.
Assessment at this stage (in addition to economics based on resource quality), should include as a minimum:

- site-specific tenement constraints or future project risks, for example subsidence under rail, road or waterways, strata title issues, property ownership etc;
- potential hydrological or hydrogeological risk associated with water ingress due to perching aquifers, surface to seam flows or associated slope stability issues in open cut mining;
- approximations of significant (mine plan constraining) geological structure from observed major RL displacements and regional knowledge;
- approximations of joint/cleat orientations from regional inferences and the associated impact on mine planning options; and
- overburden, seam and floor characteristics; more specifically rock mass and material properties and their impact on slope stability and bench orientation in open cut mining or heading stability or caving characteristics in underground mining.

Due consideration, risk analysis and sensitivity analysis of various planning options at conceptual level based on comprehensive analysis and interpretation of available data including resource quality, economic, geological and geotechnical parameters is essential to deliver:

- An assessment(s) of project risk and value that is more likely to be validated than refuted by future (downstream) exploration studies and analysis.
- Should business approval progress to pre-feasibility, an exploration program and study design can be delivered with sound logic based on the conceptual study findings and identified areas for further investigation. This can incorporate adequate and appropriate data collection and testing requirements, procedures and analysis/reporting requirements to maximise the understanding of project risks. In depth detailed team planning will almost certainly optimise exploration expenditure through prioritising exploration and analysis requirements relating to project development needs.
- Reducing the surprises in downstream project development. Getting it right here may even go a long way to delivering everyone’s ultimate goal; a mine plan that evolves into a mining operation that optimises economic return and delivers few surprises.

**KEY GEOTECHNICAL ASSESSMENTS AT THE CONCEPTUAL STAGE OF MINE PLANNING**

There are a number of key data sources that frequently exist at a conceptual mine planning stage, from which priority geotechnical assessments can easily be made and assessed in balance with other important factors, (including hydrological, gas, etc). The following presents a descriptive general approach in such assessments and includes hypothetical examples.

Geology and geotechnical inputs at concept mine level are clearly interlinked and not mutually exclusive. The scope of a concept study would clearly reflect the type of mining being considered – open cut or underground. For example an underground longwall conceptual study may include the following sections:

- coal quality (impact on reserves and various);
- geology:
  - description of target formation,
  - regional geology, structural trends and coal measure sequence,
  - specifics of exploration undertaken, exploration history and current resource status,
  - structural trends,
- intrusives,
- description of coal measures and individual seams,
- topography,
- hydrogeology, and
- seam gas;
- geotechnical considerations:
  - roof and floor conditions,
  - seam conditions,
  - stress magnitude and orientation,
  - jointing and cleating,
  - pillar dimensions,
  - ground support requirements,
  - consideration of longwall cavability,
  - multiple seam mining implications, and
  - consideration of in situ horizontal stress on mine layout.

Once relevant geological and geotechnical inputs have been scoped for the deposit and mining method being considered, analysis of each parameter is required. The following outlines some of the data and associated analysis regarded as essential to address key issues at this stage.

**Previous research, back analysis and benchmarking previous industry experience and learning**

Internet and library sources provide a ready source of publicly available information in Australia, the US, and elsewhere on everything from multi-seam mining experience and associated panel/pillar design history and methodologies, to benchmarking productivities relative to different geotechnical environments. Where appropriate and comparative, such information can be used to benchmark performance and anticipate likely ground behaviour with respect to resource and reserve assessment. This can be further used to influence downstream decisions on such factors as mining method, mine layout, and equipment selection. If possible, assess using a range of methods to achieve this, and compare and identify why different results may be derived.

In many instances when considering a conceptual mine planning study in an area not previously mined, there may be very little site-specific data relating to likely operational performance in the particular resource under consideration. In these instances however, parallels can be drawn through assessing productivity and other risks impacting operational performance, particularly when considering previous mining experience in the same seam, or in a seam with similar geological/geotechnical characteristics. This can be drawn from international experience and data. It does not necessarily have to be documented experience from a similar Australian resource as long as it can be demonstrated with confidence that the empirical comparisons are justified.

When assessing the strength of comparison with respect to geotechnical experience in comparative environments, particular parameters to comprehensively check should include, as a minimum:

- **System of mining.** Ensure that the operational data being compared derives from the same system of mining. This may sound like the obvious, however the geotechnical environment, open cut or underground, is highly sensitive to mining method. The impact of the geotechnical environment will differ subject to mining method. With any empirical comparison of mining data, this should be the first check prior to others to establish that an overall comparison is indeed valid, prior to further analysis.
• **Resource characteristics.** Check that general seam structural geological characteristics, seam thickness, seam rolling and horizon, rock mass and material strengths, and likely nature and density of seam cleat and jointing, for the resource being assessed are in the same general range as the study data being considered.

• **Stress environment.** Check that the range of cover depths and anticipated horizontal and vertical stresses are in the same general ranges for the resource being assessed as the comparative study data being considered.

### Regional geological structural trends

Regional structure can be reviewed from publicly available government sources. Aeromagnetic, satellite photos and gravity surveys may also give an insight into regional anomalies. Interpretation of geological structure over a resource area should always be checked and balanced against the wider existing regional trends and structural styles prior to more detailed structural interpretation from available exploration data.

Earlier generation seismic structural interpretation through a target area, although less advanced than more recent seismic technology, can certainly assist in structural interpretation. Often, the seismic interpretation can be enhanced through reprocessing of base information using more current technology.

Once a geological interpretation has been established, mine planning constraints, in particular based on trends, locations and displacement of faults or significant folds are normally applied.

However, in addition to fault location and displacement magnitude, assessment of the nature of interpreted geological structure with respect to orientation and dip relative to the coal seam/panel layout is important. For example, interpolated discreet near seam low-angle compressive thrust faults may have a more adverse impact (over a greater lateral extent) on strata stability, roof support requirements and potential mine planning constraints than subvertical normal faults of limited lateral extent. Structure can also have an adverse impact on roof/rib stability for both longwall and development mining.

It is largely the **nature** of the geological structure with respect to its orientation relative to longwall faces or development headings and associated dip with respect to the roof, rather than simply magnitude of displacement, that forms the major constraint with respect to mining. There now exists a number of Australian examples of demonstrated longwall retreat through significant faults, which have been achieved through appropriate hazard assessment and operational practice. Significant displacement faulting, although a risk, should therefore not automatically be a planning constraint. Care should be taken when assessing the risks associated with fault mine through related to seam displacement considered in the context of seam thickness and roof and/or floor strength. For example a +5 m seam displacement in a 1.5 m seam with strong roof and floor when fully assessed may present substantially more risk than a +5 m seam displacement in a 5 m seam with weak roof and floor.

For example a seam displacement of greater than 5 m in a 3 m thick seam with strong roof and floor when fully assessed may well present less risk than a 2 m seam displacement in a 5 m seam with weak roof and floor.

Where possible, attempting to assess the variations in rock mass characteristics associated with structures, to facilitate a more comprehensive assessment of geotechnical implications and associated mining risk, is also justified.

### Seam splitting and rider seams

The geotechnical impact of seam split areas, particularly in the near roof of underground longwall and development headings should never be underestimated. There are numerous documented examples of major roof cavity and productivity delays associated with immediate seam splitting. Seam split zones in Australian underground coal mines are often associated with:

• Channelisation of strata and associated variation in rock mass characteristics and stress distributions where rider seams diverge.

• Differential compaction features. These are often (wrongly) interpreted as low angle shear zones, although the impact can be similar but more localised. Differential compaction is a geological depositional feature associated with basin development and sinking of overlying strata into the coal formation.

• Localised seam thinning.

• Increased density of jointing in the immediate roof.

All of the above can combine to form highly variable and low strength rock mass and cohesion in the immediate roof environment which may require tailored strata management and ground support practice. Preliminary hazard plans and risk assessment should most definitely incorporate the lateral extent of interpreted seam split zones and the associated consequences with respect to both specific ground support requirements and/or mine planning constraints. Geotechnical hazard plans can be used to generate mine planning schedules zoned for variation in mining rates using appropriate de-rating factors.

### Exploration core and geophysical log signatures

It is relatively easy and appropriate to manipulate these forms of exploration data to interpret rock material composition and rock mass characterisation (using selected appropriate industry standard rating schemes), of the entire overburden section for immediate roof strata assessment and higher. Such information is particularly relevant to assessing the risks and likely requirements associated with ground control, longwall cavability characteristics, mining method, productivity, and mine sequencing.

In many instances it is possible to use existing geophysical logs or electronic LAS files, and correlate these with lithological logs. Material strength in the form of unconfined compressive strength (UCS) may be estimated if existing conversion formulae for the assessed seam in the same area to convert Sonic Velocities into UCS are available. Sonic velocity is a function of rock elasticity, and this can be correlated with rock strength. By plotting the sonic velocity for the immediate overburden to the seam, the rippability of the overburden can therefore be assessed as illustrated in Figure 3 through use of industry standard generalised rock strength correlations as illustrated in Table 2.

<table>
<thead>
<tr>
<th>Sonic velocity (m/sec)</th>
<th>Rock strength</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;1500</td>
<td>Very low</td>
</tr>
<tr>
<td>1500 - 2500</td>
<td>Low</td>
</tr>
<tr>
<td>2500 - 3500</td>
<td>Medium</td>
</tr>
<tr>
<td>3500 - 4500</td>
<td>High</td>
</tr>
<tr>
<td>&gt;4500</td>
<td>Very high</td>
</tr>
</tbody>
</table>

Such correlation facilitates estimation of the immediate roof, floor and seam material strengths. Figures 2 and 3 illustrate (relatively straight forward) LAS file manipulation to produce valid graphical output in the form of valid industry recognised geotechnical characterisations for underground and open cut scenarios.

Geophysical logs when assessed with geological (lithology logs) can be particularly useful in assessing the extent and position of any Rider seams and the extent of laminated or low strength...
roof units for likely support scenarios. Both of these factors warrant due consideration as they can have particular impacts on the geotechnical mining environment. Estimation of coal mass roof ratings (CMRRs) from available exploration data can easily be achieved through appropriate industry methodologies, and can facilitate early detailed geotechnical characterisation even at a very early stage of mine planning.

**Assessment of floor stability from available exploration data**

Weak and/or easily degradable floor may in certain instances constrain open cut or longwall mining potential, and therefore mine layout. In the case of underground development, the mining risks associated with trafficability in development headings, pillar behaviour and floor heave in such conditions warrant consideration at an early stage.

**Assessment of joint and cleat orientation with respect to mine layout**

In many instances detailed geotechnical interpretation of cleating/jointing from petroleum and gas exploration data, core logging, core orientation, and acoustic scanner information is possible. Such interpretation can assist in the assessment of optimum panel layouts with respect to roadway heading and longwall face stability.

Inappropriate panel layout with respect to cleat/joint orientation can have adverse rib stability impacts on development mining and under longwall abutment loading and can also result in unstable longwall faces. The risks and implications associated with cleat/joint orientation and density with respect to mine layout should be regarded as a priority geotechnical parameter warranting consideration in mine planning. Risk assessment of
anticipated cleat and joint orientation with respect to panel layout is therefore necessary at the earliest possible stages of mine planning.

In the assessment of open cut geotechnical mining risk, interpreted jointing/cleating should be considered in combination with preliminary rock mass and material assessment and interpreted bedding plane orientation relative to mining direction. This can be assessed through consideration of typical failure mechanisms and in general terms, the impact and extent of potential failures will be exacerbated if the discontinuities strike parallel with the pit face. A preliminary risk assessment incorporating standard potential failure mechanisms (as outlined by Hoek and Bray, 1981), from data interpretation or inference, should be incorporated at concept level, making a preliminary assessment of the following as illustrated in Figure 4.

- The potential for toppling failure from vertical/subvertical joint sets.
- The potential for planar failure due to low angle dipping discontinuities. This can present a particular problem where low angle discontinuities intersect subvertical joint sets as illustrated in Figure 4.
- Planar failure due to low angle structures intersecting subvertical joints.
- Wedge failures due to intersection of opposing discontinuity sets.
- Mass slump mechanisms in overburden soil or heavily fractured rock.

![Diagram](image.png)

FIG 4 - Preliminary open cut slope stability assessment (from Hoek and Bray, 1981).
In addition to the impact of joint and cleat orientation, both low wall and highwall stability should consider the risk associated with the following parameters relative to mining method:

- geometry, including floor dip, slope angle;
- placement sequence with respect to spoil;
- material properties including (if available or inferred) strength, shear strength, weathering, plasticity, fabric structure, saturated and unsaturated unit weight;
- floor material strength and degradability;
- identification and categorisation of discontinuities, shears or weak bands, assessment of failure potential along these surfaces and the potential for and reactivation with increased hydrostatic surcharges;
- standing water table, aquifers and general groundwater conditions; and
- blasting practice and impact on stability.

In consideration of underground mining, orientation and density of jointing and cleating can impact on the stability of the roof and rib from a geotechnical, and therefore mine planning perspective. Well developed cleating and/or jointing running near parallel to planned mining development operations will likely impact adversely on roadway rib and roof stability. Orientation of cleating relative to proposed longwall panels may also have an adverse geotechnical impact on longwall face behaviour.

Experience shows that a heading orientation of at least 20° to the cleat/joint direction is required to minimise adverse impact with respect to both roof and rib stability. However the optimum underground panel layout should be cognisant of both the predominant joint and cleat orientation, the major and minor principal horizontal stress orientations and consider the orientation of geological structural zones. Figure 5 illustrates a hypothetical longwall gateroad panel layout considering joint/cleat orientation and in situ principal horizontal stress.

![Diagram showing joint/cleat orientation and hypothetical longwall gateroad panel layout](image)

**Existing geological models**

If existing geological models are available at concept level, gains can be made from comprehensive analysis of existing geological strata models from a geotechnical perspective. In many instances the seam, as well as overburden strata is modelled in the form of a three dimensional model. Mine planning is also three-dimensional. Assessing the consistency of seam thickness and interpretation of immediate roof lithologies and overburden characteristics from the existing geological model can deliver key data which can be used for preliminary hazard and risk analysis of geotechnical parameters including:

- rock mass and material assessment of the immediate roof, seam and floor characteristics for both open cut and underground mine planning purposes;
- rock mass and material characterisation of the overburden for analysis of goaf cavability and associated impact on pillar extraction, abutment pillar loading, and longwall face performance; and
- assessing broad scale variations in dip which may pose a risk to both horizon and ground control, particularly for longwall mining.

**Stress orientation and magnitude**

Information on stress magnitude and orientation may be available from a number of sources, including coal seam hydrofraccing methods which are often commonplace in petroleum/gas field evaluation. In such instances, major principal horizontal stress magnitudes and orientation can be approximated by formula and assessed in the context of mine layout. Stress orientation may also be derived from caliper logs or acoustic scanner analysis using borehole breakout.

Such information can prove useful in assessing or testing the assumption of regional horizontal stress fields. Approximated horizontal stress magnitudes should be considered with caution, as they are entirely dependant on the modulus properties (stiffness) of the rock material being considered. Stiffer materials will inherently attract higher in situ stresses. When approximating horizontal stress magnitude from available data and assessing likely ground behaviour/reaction, it is therefore critical to make the assessment in the context of the materials being considered. Further, a number of stress domains may exist across the resource, modified in particular by intrusives and faulting. If sufficient information is available and providing like materials are being assessed and compared, approximated horizontal stress magnitudes can reasonably be compared and variations/anomalies identified over a resource. Any differences in stress orientation or magnitude (measured or predicted) over the resource may flag the potential for adjacent associated geological structural influence which may, in itself, prompt the targeting of further exploration investigation and analysis.

Assessing the impact of in situ stress orientations relative to underground development driveage and strata management requirements should take into consideration:

- an estimation of in situ vertical stress from cover depth and consideration on rib stability and support requirements;
- assessment of the regional horizontal stress field and typical horizontal to vertical stress ratios for the seam under consideration; and
- assessment of available in situ stress orientation measurements, inferences or estimations from exploration data as described above.

Assessing the impact of in situ horizontal stresses relative to longwall panel and face orientation is also an important consideration. It has previously been found (Hasenfus and Su, 1995) and continues to be observed in Australian longwall operations, that the maingate is stressed relieved when Ø, the angle between the in situ principal horizontal stress direction and the maingate orientation, is between 90° and 180°, with the best conditions prevalent at Ø = 160°. Conversely, the maingate is stress concentrated when Ø is between 0° and 90°, with the maximum concentration at Ø = −70° and negligible concentration between 0° and ∼25°. Figures 6 and 7 illustrate a model of this relationship between Ø and the relative horizontal stress increases or decreases in the maingate.

![Diagram showing impact of in situ horizontal stresses](image)
Stress notching of *in situ* horizontal stress (eg on approaching a previous goaf leading to a ‘superstressed’ situation) is an important consideration for mine planning, pillar design, and tailored secondary support requirements. The degree of impact in this situation is dependant on the orientation of maingate or tailgate and/or virgin goaf areas with respect to *in situ* principal horizontal stresses. It is important to assess the risk of unfavourable panel orientation and to consider the preference for maingate or tailgate in stress notch (if panel orientation unfavourable) and preferred direction of retreat and mining sequence, balanced against other factors.

Assessment of anticipated vertical stresses on the longwall face. Variations in vertical stresses on the immediate longwall face will be anticipated as planned longwall panels advance from shallower supercritical through critical range to deeper subcritical scenarios. Preliminary assessment of subsidence profiles at various depths at assumed angles of draw could then be estimated as illustrated in Figure 8.

The progression may not necessarily translate into increased anticipated loading on the longwall face. A critical factor in such an analysis is the likely goafing behaviour associated with overburden strata and the absolute vertical stress increase associated with the proposed panel face width. Sufficient overburden lithological data may well be available at a conceptual stage to assess (and in the case of multiple projects compare) likely face loading implications associated with longwall width taking into consideration overburden and caving characteristics.

**Lack of horizontal stress**

There are incidences of roof failures, that have been attributed to lack of confining stress, in particular where influenced by the presence of jointing. The general style of failures in these instances may be confined by parallel running joint sets and attributed to a lack of confining stress acting on the joint surfaces and therefore strata inability to maintain stability. Lack of confining stress may also be associated with proximity to geological structure (eg on the crest of seam rolls), or around faults.

The impact of potentially low magnitudes of confining *in situ* horizontal stresses and impact on the mine layout should be incorporated into hazard assessment, particularly in shallow underground environments or with limited competent material cover. In some instances, the assessed risks associated with limited competent rock cover may be of sufficient magnitude to preclude mining potential. A common rule of thumb is to maintain a minimum of 25 m to 30 m of competent material in the mining seam roof.

**Longwall caving characteristics**

Proposed longwall panel width against overburden depth ratios directly impact longwall caving characteristics, face conditions, surface subsidence profiles and chain pillar design, together with anticipated ground support requirements and productivity assumptions. A number of industry recognised empirical methodologies exist to assess estimated pillar loading, strength

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**FIG 6 - Horizontal stress notching in longwall mining (from Chekan and Listak, 1992).**

**FIG 7 - Effect of panel orientation on horizontal stresses (from Hasenfus and Su, 1995).**
and design requirements which incorporate depth of mining and face width. Empirical design methodologies and bench marking mining experiences in similar geotechnical environments utilising available geological information can be used at the conceptual level of mine planning to establish base roadway development and longwall requirements and other potential impacts. Previous pillar design experience and stress modelling from the same seam in similar mining environments should be incorporated where available.

Specific interpretations/inferences to assist in assessing likely goafing and longwall face behaviour and associated geotechnical risk can be made at preliminary mine planning level. This can be assessed through the combined influences of cover depth, overburden lithology, and joint/cleat orientation relative to the longwall. Specific initial considerations may include:

- Assessing the nature of the overlying strata with respect to rock material and rock mass strength, rock composition, and bedding plane characteristics and the potential impact on longwall face and abutment loading. A broad interpretation of overburden lithology can be made through manipulating electronic LAS files to produce geophysical plots of characteristic overburden for assessment with respect to anticipated goafing behaviour.
- The longwall panel width against overburden depth ratio will impact on the caving characteristics, face conditions and surface subsidence profiles as previously discussed.
- Joint orientation with respect to proposed longwall face orientation is regarded as potentially having a significant impact on longwall face stability and goafing behaviour.
- A potential high level of risk exists in any longwall system if longwall specification based on anticipated ground behaviour is ill considered. At conceptual level, a broad assessment of the overburden behaviour based on interpretation from (in some cases limited) exploration data and benchmarking this against behaviour in comparable environments for existing longwall operations can be undertaken.

In potentially more complex or challenging geotechnical environments, more detailed numerical modelling may be justified at a later stage of mine planning when adequate high confidence geotechnical parameters can be established from exploration test work. This is likely to assist in validating empirical assumptions with respect to goafing and loading behaviour made at concept level.

**Pillar design assessment**

Industry recognised and current empirical pillar design methodologies (eg UNSW, various ALP based methodologies) can be undertaken at the conceptual level of study to gain an appreciation of likely mine pillar requirements based on available input data and parameters. With limited available input data this approach in general is justified at conceptual level. In the later stages of mine planning, more sophisticated measures such as numerical modelling may be used. In any geotechnical design there is value in applying and comparing separate methodologies based on available input parameters, rather than use of simply one or other methodology. This provides a check on the validity of the design tool used specific to the resource characteristics, highlights any variations and sensitivities associated with site specific input parameters and design formulae used, and provides a more considered and auditable design process. Particular care should be taken in adequately assessing the quality and sensitivity of input parameters in any geotechnical design process used.

**Multiple seam mining implications**

Interactive problems due to stress redistributions in multiple seam longwall operations, particularly due to transfer of stress from overlying gateroad pillars to underlying gateroad pillars where superimposed, or to the underlying longwall face where superpositioned (Figure 9), can have an adverse impact on longwall face strata control or pillar performance, unless
appropriately considered and designed for in the mine planning process. Gale (2004), has recently completed an ACARP study reviewing overseas data relating to empirical experience and undertaking geotechnical modelling work in multi-seam longwall environments. From this work, Gale indicates that in general offset compared with superimposed layouts may be preferable in Australian conditions and certainly from the perspective of subsidence minimisation. The risk of adverse longwall face control under overlying chain pillars should, however, not be under-estimated.

In a case study conducted by Chekan and Listak (1992), concentrating on pillar design considerations for underlying superimposed pillars (based on ALPS pillar design methodologies calibrated with modelling), it was concluded that the two most important parameters influencing the proportion of abutment stress transferred from the upper to the lower mine pillars (referred to as the multiple seam factor – MSF) were, in order of sensitivity, interburden thickness followed by pillar width. Pillar length was found to be a far less sensitive parameter. This study was based on three and four heading gateroad scenarios.

In a hypothetical situation, assuming an interburden thickness between superimposed pillars of 50 m (165 ft) and upper pillar sizes of approximately 100 ft (30 m), the USBM studies (Figure 10) indicate an approximate MSF of around 30 per cent. That is, 30 per cent of the calculated abutment load from the upper pillars can be anticipated to be transferred to the lower pillars in this situation. Although specific to American pillar design calculations (ALPS) and multi-seam longwall mining conditions for three heading gateroads, and also calculated for smaller pillar sizes, the example none the less serves to illustrate that, where the interburden between seams is less than 50 m, there is likely to be a component of load transfer requiring that can be estimated and considered further in designing chain pillars for superimposed panels.

More recently Ellenburger, Chase and Mark (2003), NIOSH conducted an empirical study into case histories involving undermining previous longwall panels involving 12 different coal seams with seam heights ranging from 1.2 m to 2.1 m and overburden thicknesses ranging from 75 m to 620 m. A strong empirical relationship was established between the amount of damage to the lower seam caused by load transfer from the upper seam, and the overburden to interburden ratio (Figure 11).

The US database study concluded the following:

- No significant damage to the lower seam was recorded when the overburden-to-interburden (OB/IB) ratio was less than approximately seven.
It is possible to successfully mine, even at high cover and with large OB/IB ratios, when the mining is carefully planned to take place in the stress shadow beneath fully extracted goaf areas.

In summary, there is a need to not over generalise and to recognise the complexities associated with stress redistributions in multi-seam mining operations specific to local conditions, mining timing/sequence, local geotechnical parameters, and in the context of what the mine design is trying to achieve. Nonetheless, at conceptual level with limited data and in the absence of a record of mining history, assessing mining experiences in comparable geotechnical environments using published data may deliver a valid and logic based assessment of likely behaviour in a multi-seam environment. Given local specific conditions however, further assessment which may take the form of geotechnical modelling may be warranted in downstream mining studies when sufficient high confidence input data is available to validate initial assumptions and interpretations made regarding stress interactions.

Subsidence considerations

A number of alternative approaches to subsidence prediction are available, using empirical or mathematical relationships. At conceptual mine planning level, the primary purpose of this evaluation may be in regard to environmental impacts, an assessment of the further requirements of mining approvals, or to assess the potential lateral impacts on adjacent lease ownerships and associated mine layout constraints.

Analysis at conceptual level should include:
- review and back-analysis of previous regional subsidence history;
- determination of approximate subsidence magnitudes and lateral influence for the proposed mine layouts;
- potential impact with respect to perched aquifer breaching and associated inflow;
- generation of post subsidence surface contours across the proposed mining area (if sensitive and required); and
- a preliminary assessment of potential subsidence impacts and recommendations for further study should the project progress. Typical mitigation and remediation measures (including design, and pre/post mining) may be included at this stage.

A number of subsidence predictive tools, including for example empirically derived subsidence curves (eg Holla, NCB), can be used as a tool to complete analysis. However care should be taken to select the most appropriate method for the seam environment being considered. A second check analysis using a separate methodology may be warranted at this level depending on the level of mine planning sensitivity and risk in relation to projected subsidence.

A REPORTING FRAMEWORK FOR GEOTECHNICAL CLASSIFICATION OF MINE PLANNING PROJECTS

As previously outlined, there are clear input requirements for effective project valuation at various stages of the mine planning cycle. The author has argued the case for comprehensive...
The value of early geotechnical assessment in mine planning

Table 3 illustrates the proposed data interpretation requirements at various stages of geotechnical categorisation, from implied to verified. Although focused primarily on metals orebody assessment, such a framework specific to coal could provide mining and financial Institutions with a guide to the level of geotechnical input required for a project at any particular stage of development.

From the perspective of geotechnical risk sensitivity in the process of mine planning and project development, the author raises the following questions to the industry in search of debate and feedback:

1. How well are resources currently assessed in mine planning and during project development, particularly at the early stages of assessment, from the perspective of geotechnical risk, relative to other key drivers including coal quality and valuation? How sensitive is such assessment in determining the success or otherwise of a project?

### Table 3

Example proposed reporting framework for geotechnical projects (from Haile, 2004).

<table>
<thead>
<tr>
<th>Data type</th>
<th>Implied</th>
<th>Qualified</th>
<th>Justified</th>
<th>Verified</th>
</tr>
</thead>
<tbody>
<tr>
<td>General requirements and geotechnical model reliability</td>
<td>No site-specific geotechnical data necessary</td>
<td>Project-specific data are broadly representative of the main geological units and inferred geotechnical domains, although local variability or continuity cannot be reliably accounted for.</td>
<td>Project-specific data are of sufficient spatial distribution (density) to identify geotechnical domains and to demonstrate continuity and variability of geotechnical properties within each domain</td>
<td>Site-specific data are derived from local in situ rock mass.</td>
</tr>
<tr>
<td>Geological model</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Stratigraphic boundaries</td>
<td>Inferred from regional geology</td>
<td>Reasonable knowledge of major units and geometry</td>
<td>Well constrained in the vicinity of the mine excavations and infrastructure</td>
<td>Mapped in the field</td>
</tr>
<tr>
<td>Weathering/alteration boundaries</td>
<td>Inferred from regional geology</td>
<td>Based on geology model</td>
<td>Well defined grading of weathering and local variability</td>
<td>Mapped in the field</td>
</tr>
<tr>
<td>Major structural features</td>
<td>Inferred from regional geology</td>
<td>Major ‘dislocations’ interpreted</td>
<td>Drilling sufficient to be well constrained in continuity, dip and dip direction</td>
<td>Mapped in the field</td>
</tr>
<tr>
<td>Rock mass data</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Discontinuity</td>
<td>Based on general rock type characteristics</td>
<td>Estimates of RQD/FF and number of defect sets from resource data (will probably contain directional bias)</td>
<td>RQD/FF statistics and number of defect sets representative of all geotechnical domains and directions</td>
<td>Multi directional FF from in situ mapping and visual count of defect sets</td>
</tr>
<tr>
<td>Intact material strength/ deformation characteristics</td>
<td>Based on general rock type characteristics</td>
<td>Field estimates</td>
<td>Field and laboratory estimates and variability</td>
<td>Field and laboratory estimates</td>
</tr>
<tr>
<td>Defect data</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Orientation</td>
<td>Inferred from regional geology</td>
<td>Orientation inferred from geological model</td>
<td>Dip and dip direction statistical data from drill holes.</td>
<td>In situ measurement of dip and dip direction from excavation mapping.</td>
</tr>
<tr>
<td>Surface characteristics</td>
<td>Estimated on precedent experience</td>
<td>Estimated on precedent experience</td>
<td>Statistical estimates from core logging for all defect sets. Laboratory shear strength testing of critical defects.</td>
<td>Statistical estimates from in situ measurements. Laboratory shear strength testing of critical defects.</td>
</tr>
<tr>
<td>Volumetric distribution (continuity and spacing)</td>
<td>Estimated on precedent experience</td>
<td>Estimated on precedent experience</td>
<td>Estimated on precedent experience</td>
<td>Persistence and spacing measurements</td>
</tr>
<tr>
<td>Stress regime</td>
<td>Mean regional trend</td>
<td>Local magnitude and orientation based on local experience or modeling</td>
<td>Measured or inferred from in situ performance</td>
<td></td>
</tr>
<tr>
<td>Seismicity/earthquake</td>
<td>Based on general experience</td>
<td>Based on geology model</td>
<td>Based on regional trends</td>
<td>In situ experience</td>
</tr>
<tr>
<td>Geotechnical model/ domains</td>
<td>Based on geology model</td>
<td>Based on geology model</td>
<td>Based on geotechnical data.</td>
<td>Based on in situ data</td>
</tr>
<tr>
<td>Hydrogeological model</td>
<td>Based on general experience</td>
<td>Hydrogeological study</td>
<td>Local observations/ measurements</td>
<td></td>
</tr>
</tbody>
</table>
2. Given the traditional role and required (defined) competencies of persons traditionally used to assess a project with respect to resource and reserve definition generally to JORC Code guidelines, is there a real justification for the involvement of experienced geotechnical practitioners and more defined input at the various process levels?

3. In view of both the above factors, are there reasonable grounds for developing a reporting framework, which can be used as a guideline for geotechnical classification of mining projects, specific to coal, which could prove beneficial to resource companies?

REFERENCES


Coal Pillar Design Criteria for Surface Protection
D Hill

ABSTRACT

Large areas of ground are permanently supported on coal pillars, both in extensive old workings and current drivages in active mining operations. Continued growth of civil infrastructure is resulting in more surface development above old workings, commencing with an outline of the most widely accepted empirical Factor of Safety (FoS) methodology. Building on this empirical foundation, criteria are derived that facilitate rational pillar design in circumstances involving protection of surface infrastructure or other sensitive features.

INTRODUCTION

Pillars serve two main roles: promoting the serviceability of underground roadways adjacent to areas of extraction (eg longwall chain pillars) and maintaining long-term regional stability (eg main heading pillars). These pillars are an operational constraint determining the amount of roadway development required. As such, the general need is to minimise pillar widths wherever possible, noting that overly-large coal pillars do not result in significant improvements in serviceability or enhanced regional stability. On the other hand, inadequately-sized pillars can cause major operational difficulties and large-scale rock mass instability, which may be manifested as discernible surface ground movement (ie subsidence), with impacts on other stakeholders.

Over 200 years of underground coal mining in Australia has resulted in large areas of ground supported on coal pillars, including very extensive old workings in generally inaccessible redundant mines and current drivages in active mining operations. Also, continuing growth (in terms of both size and complexity) of the civil infrastructure is resulting in more surface development above old bord and pillar mines, as well as the increasing need for mine development beneath existing, frequently sensitive, surface structures. The result is greater possibility of conflict between miners, developers and regulatory bodies, with the potential for sterilisation of underground coal resources and/or escalating surface development and infrastructure protection costs.

One positive factor has been the significant improvement in the general understanding of coal pillar behaviour and stability over the last 40 years, in Australia as well as overseas. This paper examines some of the issues to be considered when undermining surface structures or undertaking surface development above old workings.

The Factor of Safety (FoS) methodology widely employed for the assessment of pillar stability is reviewed, including the key geometrical, geological and statistical concepts associated with the probability of pillar failure; local and international experiences are examined and significant parameters isolated. Common concerns are addressed in the context of actual practical experience, utilising a risk management approach. Recent advances in methods for the assessment of pillar stability are put forward, along with criteria for arriving at rational design outcomes.

FACTOR OF SAFETY METHODOLOGY

The empirical coal pillar Factor of Safety approach is considered to represent the most reliable methodology available for analysing the long-term stability of regular arrays of pillars that are wide with respect to cover depth. Alternative numerical approaches are hampered by our inability to accurately define rock mass properties and develop constitutive laws that fully define rock mass behaviour. The inherent variability of the underground rock mass (and specifically coal measures strata) is also a challenge, in that system failure is very often associated with an anomaly that may be particularly difficult to model.

The FoS approach involves back-analysing case histories (ie failures and successes) to derive a means of estimating coal pillar strength. The FoS is simply the ratio of pillar strength (S) to applied load (L). The great merit of this empirical approach is that it utilises full scale, four-dimensional models (ie coal mines). The methodology draws inferences directly from reality, whereas the alternative numerical approaches draw inferences from simplified simulations of reality.

Essentially, empirical approaches facilitate the derivation of a probability of success in a particular situation, based on the analysis of prior successes and failures (ie intact and collapsed panels of pillars). Limitations of empirical approaches can be associated with the nature (ie the size and quality) of their underlying databases. Difficulties may arise when an empirical relationship is employed in a situation beyond the experience quantified in the database. The compilation of reliable and relevant databases is a key consideration, as is their subsequent upkeep and extension.

With specific regard to general coal pillar design in Australia, the formulae developed in recent years by the UNSW (Salamon et al, 1996, Galvin et al, 1998) are considered to represent the current state of the art in empirical FoS approaches. The formulae are founded on extensively researched and broadly-based databases of mining experience. These formulae represent the culmination to-date of work commenced some 40 years ago in South Africa after the 1960 Coalbrook disaster (Salamon and Munro, 1967). A combined Australian and South African database has been applied to the derivation of formulae that are considered widely applicable.

The range of parameters within the UNSW combined failed and intact pillar database can be summarised as follows:

- depth: 20 m to 510 m
- mining height: 1.0 m to 9.2 m
- smallest pillar dimension: 2 m to 32 m
- bord width: 3.7 m to 15.0 m
- percentage extraction: 30 per cent to 90 per cent
- width to height ratio: 0.9 to 11.2
- time to failure: 0 to >80 years

The FoS derived using the UNSW formulae can be related directly to the probability of stability, as illustrated in Figure 1. Assuming full tributary area loading, it can be seen from the figure that a probability of stability of 99.9 per cent is attained at a Factor of Safety of 1.63. Further increases in FoS have diminishing effect, as the stability curve asymptotically approaches 100 per cent. Increasing the FoS is therefore not always the most effective response from a risk management perspective, given that the probability of failure can only be reduced by <0.1 per cent.

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The consequences of failure are a key consideration, as these determine an acceptable probability of pillar failure, which in turn allows an appropriate FoS to be determined. Prudent risk management suggests that the probability of failure for long-life pillars beneath sensitive surface features should be negligible.

In Australia, long-life critical pillars (eg in main headings and for surface protection) are often designed to a minimum FoS of 2.11, which equates to a failure probability of one panel in a million, based on the power law strength equation developed by the UNSW (Galvin, Hebblewhite and Salamon, 1999). This reduces the likelihood of instability to a level that would be considered acceptable in other fields of public interest. Similar criteria are applied in South Africa, where the formulae originated (Salamon and Oravecz, 1976).

Further consideration of the nature of pillar loading is generally required for panels that are narrow with respect to depth (ie typically at panel span to depth ratios of <1). The assumption of full tributary area loading can significantly overstate pillar load in these circumstances, resulting in highly conservative and in some cases inappropriate designs. There is widespread industry experience of the stability benefits of reduced panel spans (eg in the design of main headings pillars in the Southern Coalfield and with narrow ‘stress relief’ pillars adjacent to longwall installation roads).

Provided that workings are designed to appropriate Factors of Safety, it is necessary to look beyond this concept to obtain any further assurance of stability that may be required. Additional factors that may require specific attention include:

1. pillar width to height ratio,
2. future pillar loading history,
3. the nature of the roof and floor,
4. the presence and impact of weak bands/discontinuities in the pillars, and
5. long-term pillar behaviour.

**PILLAR WIDTH TO HEIGHT RATIO**

The role of increasing width to height ratio in promoting enhanced pillar stability has long been known. Back analysis of case histories from around the world has shown that width to height ratio exerts a major influence on coal pillar strength. At low ratios (<3) overloaded pillars tend to fail in a brittle, uncontrolled fashion, whereas at higher ratios (>4) the coal pillars demonstrate a more plastic form of deformation: significant displacement may take place in the form of convergence of the roof and floor, as well as rib spall, but the pillar core remains confined and tends to retain its load carrying ability, generally without failing in the commonly understood sense.

This was illustrated by Das (1986) in tests on Indian coals, see Figure 2. It was also shown by Madden (1987) with tests on sandstone discs during the development of the squat pillar formula (he used sandstone because coal samples are more heterogeneous and difficult to prepare), see Figure 3. It is noteworthy that the shapes of the stress-strain curves are similar at equivalent width to height (w/h) ratios for the two materials.
International industry experience confirms the importance of width to height ratio to pillar stability. Incidences of collapse are concentrated at low ratios, see Figure 4.

Width to height ratio, applied in conjunction with other criteria (eg FoS), is a useful indicator of design reliability. This is illustrated in Figure 5 (Hill and Buddery, 2004), which presents the FoS versus pillar w/h ratio relationship for a combined database of failed South African and Australian bord and pillar panels, plus a database of highwall mining (CHM) failed pillar cases (UNSW, 1995; Madden and Hardman, 1992; Strata Engineering, 2001).

The three databases are complimentary in nature, reflecting the range of experiences of their respective industries. For example, the Australian data provides insight with regard to pillar behaviour at relatively high w/h ratios and furnishes the failed case at the w/h ratio of 8. In contrast, the South African coal industry has traditionally been characterised by geometries involving lower w/h ratios, which is partly reflected in the maximum w/h ratio of only 3.7 for a South African failed case. Similarly, CHM pillar cases cover the lower end of the range of w/h ratios, from 0.6 to 1.4.

There are no failed cases in this combined South African and Australian database with a w/h ratio of greater than 8, even at a very low Factor of Safety, and there is only one failed case at a w/h ratio of greater than 5. The highest Factor of Safety assigned to a bord and pillar collapse is 2.1 and this was associated with a w/h ratio of only 2.2. Although there are failed CHM pillars with Factors of Safety of >2, all of them have pillar width to height ratios of <2.

A limit envelope can be defined for the database of failed cases, illustrated by the curve and given by the following equation:

\[ \text{w/h ratio} = 22.433e^{-1.1677 \times \text{Factor of Safety}} \]

Beyond this envelope, there is no precedent for failure within these databases. It is worth noting that the exclusion of the CHM pillar data would not materially change the shape of this limit envelope.

In the case of long life (>5 years) pillars, if it is reasonable to assume that the panel is, or will at some point in the future, be subjected to full tributary area loading, then it is generally considered prudent to design outside the envelope defined by this equation, even though there are many examples of stable pillars that fall within it.

Furthermore, in the case of important long-life pillars (eg main headings and barriers), it is considered prudent to allow an additional margin beyond this envelope. A margin of 20 per cent is the generally suggested minimum, which is defined by the second (ie outer) curve in Figure 5 and the following equation:

\[ \text{w/h ratio} = 26.919e^{-0.973 \times \text{Factor of Safety}} \]

In the case of pillars required for the permanent protection of critical surface features/structures, an ongoing broader (ie global) review of coal pillar behaviour suggests that even in extreme circumstances involving unusually weak floor, coal and/or roof that the potential for failure can be effectively excluded by designing to a minimum Factor of Safety of 2.11 (ie a failure probability of #1 in a million), coupled to a minimum width to height ratio of 5. Note that in this context, 'failure' means pillar collapse due to the failure of any element (ie roof, floor or the pillar itself) in the overall structural system. The issue of long-term pillar behaviour is addressed later in this paper.
FUTURE LOADING HISTORY

If the pillars are to be subject at some point to stress increases due to ongoing mining activities (ie abutment loading), it is usually the case that a design to a higher FoS will be undertaken. In South Africa, for example, pillar extraction workings are generally designed to a minimum FoS of ~2.

Inadequate coal pillar design associated with uncertainties and inaccuracies regarding the determination of abutment loads adjacent to extraction areas has been associated with a number of cases of instability, in Australia and overseas. This puts an emphasis on the understanding of system stiffness, the design of barrier pillars and overall panel geometry, including panel width to depth ratio.

There are circumstances that may have potentially positive impacts on the future pillar loading conditions, such as buoyancy effects associated with the gradual increase in water level and eventual flooding of old workings.

ROOF AND FLOOR PROPERTIES

The South African and Australian databases from which the UNSW coal pillar design formulae have been derived cover a broad range of roof and floor materials, including mudrocks, coal, siltstone and sandstone. Therefore, these materials and the variability in coal pillar strength that may be associated with them are implicitly recognised and catered for within the Factor of Safety approach.

The uncertainty associated with the natural variability in coal measures strata usually prohibits design to low Factors of Safety (eg a FoS of 1.01 is generally unacceptable, even though strength nominally exceeds stress). Geological variability partly accounts for the scatter in the failed pillar cases population and necessitates design FoS values of typically >1.5, equivalent to very low probabilities of failure.

Pillar failures historically associated with weak floor can often be explained in terms of the criteria outlined previously (notably FoS plus w/h ratio). Even in known very weak floor environments, incidences of pillar collapse are again concentrated at low w/h ratios, see Figure 6.

Nonetheless, specific consideration should be given to the application of these design formulae in the presence of extremely weak roof and floor materials. In Australia, the Awaba Tuff (a claystone unit in the floor of the Great Northern Seam) has warranted particular attention. This unit tends to deteriorate in the presence of moisture.

DISCONTINUITIES/WEAK BANDS

The potential impact of discontinuities (ie localised structural defects), such as faults, diminishes rapidly as the width to height ratio of the pillars increases. This is shown schematically in Figure 7. Similarly, the influence of weak bands decreases as their aspect ratio (length/width) increases with increasing pillar width.

Again, the database encompasses pillars in a significant number of seams in different geotechnical environments; consequently the existence of pillar weaknesses is largely reflected and implicit within the variability in the failed and intact pillar cases, such that these weaknesses are very largely catered for by adopting appropriate FoS values.

Cases in which the competency of the coal seam is specifically regarded as a critical issue are rare and there are none known of in Australia.
LONG-TERM PILLAR BEHAVIOUR

The issue of the potential for long-term deterioration of workings leading to failure is an important consideration with regard to surface protection and can be addressed in the context of industry databases. It can be seen from Figure 8 that the great majority of pillar collapses occur within a short period of mining. In the Australian and South African databases, apart from one uncertain Australian case history (ie at between 80 and 170 years) the maximum recorded time interval between mining and subsequent failure is 52 years and the median time to failure is four years. Experience from the USA is generally consistent with this, even in unusually weak floor conditions.

Expressed in the context of pillar FoS and w/h ratio values, it can again be shown that the likelihood of failure reduces with time. Referring to Figure 9, it can be seen that after an elapsed period of 20 years, there are no cases of pillar collapse at FoS values of >1.5. After 40 years, there are no failed cases at FoS values of >1.4; after 80 years no failures at an FoS of >1.1.

Referring to Figure 10, it is seen that after a period of ten years, there are no cases of collapse involving pillars with width to height ratios of >3. After 40 years, there are no failed cases at w/h ratios of >2.

The industry databases strongly suggest that the majority of failures occur within a short period of mining, due either to inappropriate design or some local anomaly. As time progresses, the actual likelihood of failure decreases and those collapses that do occur involve pillar designs that would be considered increasingly marginal. There is no evidence to suggest that failure becomes inevitable or even more likely over time. On the contrary, the historical data suggests that pillar deterioration (eg associated with spall and weathering) tends to a limit over time.

SUMMARISED DESIGN CRITERIA BASED ON FoS AND W/H RATIO

As discussed, pillar design criteria should reflect the specific requirements and nature of the workings (eg short-term production panel, as opposed to long-life pillars with critical surface protection constraints). Pillar design should also give consideration to panel span versus depth, system stiffness and the nature of the loading environment. Based largely on the preceding considerations, the general approach suggested by Strata Engineering for pillars subject to full tributary loading can be summarised as follows:

1. short-term production panels, with considerable local knowledge: design may be within the failed pillar database limit envelope, under controlled circumstances;
2. short-term production workings (general): designed on the basis of being beyond the failed pillar database limit envelope;
3. key underground workings, such as main headings, with medium to long-term serviceability requirements: design on the basis of the limit envelope plus 20 per cent (ie the outer database curve); and
4. underground workings beneath critical surface structures and/or features (eg key infrastructure, such as railways/waterways): design on the basis of a minimum w/h ratio of 5 (ie ‘squat’ pillars) with a minimum nominal FoS 2.11 according to the UNSW 1998 formulae (ie a nominal probability of failure of one in a million).
FIG 9 - Time to collapse versus factor of safety.

FIG 10 - Time to collapse versus pillar width to height ratio.

FIG 11 - Design criteria for bord and pillar workings based on factor of safety and pillar – width to height ratio.
It remains important that specific attention be given to the local mining/geotechnical environment, including historical experience of pillar behaviour in the particular seam under consideration. The above criteria are only guidelines. The net effect of adopting these guidelines is as illustrated in Figure 11.

CONCLUDING REMARKS

A range of issues relevant to long-term pillar stability have been outlined. These are relevant to both the mining operation and any party involved in surface protection and development.

The derivation of Factor of Safety (‘FoS’) and an associated nominal probability of failure using appropriate formulae and input values is fundamental. It should also be evident that a range of additional criteria can be used to supplement the FoS-based assessment and improve the overall understanding of the potential for instability and the reliability of a design; in this regard, minimum pillar width to height ratio is a key parameter.

Apart from pillar strength parameters, factors that influence pillar load often warrant site-specific consideration. This paper has focussed on pillar design considerations in a loading environment that can reasonably be approximated by the full tributary area concept.

This paper has not considered subsidence mechanisms or the consequences thereof. Detailed site assessment should consider the nature of potential subsidence, including the mechanisms, magnitudes and strains associated with ground displacements. The mode of ground movement may not always be a function of pillar collapse per se. For example, at shallow depths, the propensity for ‘sink hole’-type subsidence, associated largely with intersection collapse in weak roof conditions, increases markedly.

Finally, it should be evident that there is considerable international experience of coal pillar design and stability issues, with a high level of commonality. Combined pillar FoS and w/h ratio-based design criteria have been put forward, capturing this global experience. The derivation of the next generation of design tools should aim to build on this broad experience base.

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Numerical Modelling of Undermined River Valleys — A Case Study

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ABSTRACT

Ground subsidence due to mining is a common dilemma confronting the underground coal mining industry. The effects of underground longwall mining on river valleys have come under scrutiny, especially when mining is located underneath catchment areas. As public awareness of mining impacts on the environment increases, there is the need to develop damage mitigation strategies. Results of numerical modelling show that river valley response to underground longwall mining can be simulated and evaluated.

INTRODUCTION

Various methods are used to predict closure, upsidence and compressive strain caused by valley buckling, and regional horizontal movements due to redistribution of in situ horizontal stresses around a mining area (Waddington Kay and Associates, 2002). While these predictive methods are useful, there is also a need to appreciate the deformation mechanics leading to valley base failure, i.e. why do some valleys base fail yet others do not, even when the same amount of closure has occurred? What factors determine the magnitude of vertical and lateral failure in river valleys?

The aim of this project is to develop a set of guidelines to be used in the construction of a numerical model that can be implemented to assess underground longwall mining impacts on river valleys. This paper discusses the key elements in the construction of such a model with the explicit finite difference program FLAC 2D V4. A field site, WRS1, was nominated as a suitable validation site for the study. This site is located on the Waratah Rivulet, Helensburgh NSW, and has been undermined by Metropolitan Colliery. The site contains rock bars that were previously unaffected by longwall mining.

SITE DESCRIPTION

Metropolitan Colliery is located in Helensburgh, approximately half hour drive north of Wollongong and is mining the 3.2 m thick Bulli seam. The longwall panels are 140 m wide with 35 m wide chain pillars. The colliery is currently undermining the Waratah Rivulet, which is a major tributary to the Woronora Reservoir.

The rock bar, nominated ‘WRS1’ is located in the base of a valley and has been chosen as the site to study the effects of underground coal mining on valley bases. The WRS1 rock bar is located approximately 130 m from the maingate edge of Longwall 9 (Figure 1).

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![Fig 1 - Location of WRS1 (Mills, 2002).](image)
RIVER VALLEY DEFORMATION AND FAILURE

According to Waddington and Kay (2002), river valley deformation is comprised of three main components. These are:
1. horizontal stress redistribution,
2. valley bulging, and
3. valley base failure.

Horizontal stress redistribution

Generally, in situ horizontal stress is greater than the in situ vertical stress. This phenomenon is not uncommon, and seems particularly pronounced in the Southern Coalfields.

When an opening is extracted under a valley (Figure 2):
1. The pre-existing horizontal stress is redistributed around the opening, concentrating above and below the opening.
2. This in turn increases the vertical stress, which causes the roof and floor of the opening to fail.
3. The previously redistributed horizontal stress, is displaced once more, and must find somewhere to go. As the extracted opening is in the process of creating a caved zone, the horizontal stress travels upward until it finds competent strata. This competent strata is usually located between the caved zone and the surface.

As a result, the pre-existing horizontal stress near the surface will be increased due to the redistribution of horizontal stresses due to mining. This implies that at the surface, the strata is compressed horizontally from the increased horizontal stress and expands vertically due to lack of confinement at the surface.

Valley bulging

Valley bulging is the term given to describe the inward movement of valley walls and bulging of the valley base. Valley bulging is a natural phenomenon but is accelerated as valleys are undermined or approached by underground mining. During valley formation, there is a redistribution of the in situ horizontal stress in the valley walls. The displaced horizontal stress is transferred to the valley base, causing upward movement of the base, as there is no confinement (Figure 3).

Valley base failure

With regards to valley base failure, it is known that the maximum compressive stress occurs in competent strata close to the surface. When the valley is undermined or approached by mining, the increased horizontal stress may be sufficient enough to fail the surface strata. As a result, failure progresses downward until equilibrium is reached. Figure 4 illustrates three types of failure, which are:
1. buckling,
2. wedging, and
3. shear on low angle discontinuities.

The upward buckling of the surface strata creates voids below the surface, which may impact on the hydrological aspects of rivers or creeks in valley bases. Lastly, failure of the valley base allows the valley walls to relax, resulting in closure. Due to this closure, tension cracks may be observed on the valley shoulders (Figure 3).

![Horizontal stress redistribution diagram](Image)

**FIG 2 - Horizontal stress redistribution (Waddington, Kay and Associates, 2002).**
THE NUMERICAL MODEL

Model geometry
The model geometry is based on the coordinates of survey line 3 (Figure 5), which traverses the rock bar and is used for evaluating horizontal and vertical movements. The model geometry was generated by extending the survey line, in conjunction with topographic maps, to obtain a more complete surface profile. Consideration was given to extending the line to adjacent topographic highs but the resultant geometry would be too large in terms of model building and run times. The final model geometry was determined after a parametric study, which varied the extension of the survey line and assessed whether element failure exceeded model boundaries (Figure 6).

Material properties
A series of triaxial tests were carried out on core samples from two boreholes (one vertical and one angled) at WRS1 to determine geomechanical properties. These properties have been incorporated into the model and a parametric study with 25 per cent, 50 per cent and 75 per cent reduction in strength and stiffness properties has also been carried out.

Constitutive model
FLAC 2D V4 offers a variety of plastic and elastic constitutive models. At the moment, the constitutive models of interest are:
1. Mohr Coulomb,
2. ubiquitous joint,
3. strain softening, and
4. bilinear strain hardening/softening ubiquitous joint.

The ubiquitous joint model is based on the Mohr-Coulomb failure criterion, but one problem associated with the Mohr-Coulomb constitutive model is the absence of strain softening, which cannot be ignored with a material like rock. However, this can be overcome with the strain softening and bilinear strain hardening/softening ubiquitous joint models but
either model is complex and time consuming to set up, although they may be considered at a later date. The constitutive model chosen for the initial WRS1 model is a modified version of the ubiquitous joint model, which simulates instantaneous strain softening by resetting cohesion and tension to zero in the event of element failure. If required, discontinuities can also be incorporated with minimal effort.

**In situ stress initialisation**

The formation of the river valley was expected to play a significant part in the resultant in situ stresses. Ideally, excavating the valley in stages and then cycling the model to equilibrium after each excavation would have been preferred. This option was not possible because it was felt that the model would have to encompass the entire valley and surrounding topography, instead of focussing on the immediate area around WRS1. It also raises concerns about model runtimes and grid sizes. It was decided to excavate the entire valley and then cycle the model to equilibrium in conjunction with the chosen in situ stress regime.

In situ stress measurements conducted by Strata Control Technology Pty Ltd, have indicated that the magnitude of horizontal stress is double the vertical stress at a depth of ten metres, with principal stress magnitudes of 2.1 MPa, 1.4 MPa and 0.9 MPa (vertical) (Mills, 2002). Whilst it may be simplistic
to utilise a ratio \((k)\) of horizontal to vertical stress throughout the entire model, Pells (1993) suggested the use of such a ratio for depths of up to 100 m.

In the model, a user defined function that calculates horizontal stress based on the given \(k\) ratio is used. A parametric study was carried out with \(k\) values of one, two and three respectively. Pore pressure has not been incorporated into the model.

**Valley closure simulation**

Valley closure from undermining was simulated by applying a loading velocity at the model boundaries (Figures 7 and 8). It was found that a loading rate of \(1 \times 10^{-4}\) m/s on both sides was a sufficient rate of loading without shock loading the model and causing premature failure.

**NUMERICAL MODELLING RESULTS**

A parametric study has been carried out to study the effects from varying parameters of \(k\) ratio, reduction factors for material properties and loading type. The results have been compared with expected behaviour, not field observations.

**Effect of \(k\) ratio**

It was found that when the valley was excavated and the model was cycled to equilibrium, valley closure and upsidence occurred (Figure 9), which is consistent with movements resulting from valley formation. This behaviour was observed for all values of \(k\) ratio tested. Also, the nature of the resultant stress distribution showed basic agreement with theoretical explanations with a concentration of horizontal stress at the base of the valley using a \(k\) ratio of two (Figure 10).
Effect of reduction factors

As expected, variations in the reduction factors yielded considerably different results with valley bulging and valley base failure occurring at an earlier stage when higher reduction factors were applied.

Effect of loading type

It was found that 273 mm of lateral displacement on each side was the maximum allowable displacement required to initiate valley base failure, without failure propagating and contacting the fixed boundaries.

When loading type 1 was implemented, it was found that failure occurred in the base of the valley and then propagated downwards and outwards. It must be noted that failure in general did not exceed 6 - 10 m below the base of the valley (Figure 11). It was also noted that 15 m below the base of the valley; the strata appeared to dilate, with strata above this point moving upward and strata below this point moving downward. This is in agreement with Figure 3.

When loading type 2 was implemented, and it was found 492 mm of lateral displacement was required to produce element failure in the model without boundary interference. The pattern of failure was quite different, with failure occurring at the bottom of the model, with insignificant valley base failure (Figure 12).
SUMMARY

It was found that the numerical modelling of the WRS1 field site would be best achieved by implementing a model with the following attributes:

- model geometry as illustrated in Figure 6,
- modified ubiquitous joint constitutive model,
- 50 per cent reduction in strength and stiffness properties,
- horizontal to vertical stress ratio \((k)\) of two,
- loading type 1, and
- loading rate of \(1 \times 10^{-4}\) m/s.

Overall, the results from this study have indicated that numerical modelling with FLAC 2D V4 is capable of replicating theoretical behaviour with respect to in situ stress initialisation, valley bulging and valley base failure.

MODEL IMPROVEMENTS

The following key aspects have been identified for improvement:

- model geometry,
- incorporation of discontinuities,
- constitutive model, and
- loading type.
Model geometry

The original model geometry contains several ‘blocky’ steps in the valley sides. In order to reduce potential stress concentrations, the valley sides have been refined (Figure 13). This refined geometry is also a more accurate representation of the field site, smoothing out the lack of detail generated by topographic maps.

Incorporation of discontinuities

Ubiquitous Joints (Figure 14) and Interfaces (Figure 15) both represent a plane of weakness within a material, but the key difference between the two is that an interface consists of two boundaries separated by null zones.

The advantages of modelling weak planes using ubiquitous joints is that:
1. they are simple to incorporate into the model, and some constitutive models have ‘inbuilt’ ubiquitous joints; and
2. the joint properties can be derived from triaxial testing with angle core, but this is rather simplistic.

The disadvantages of using ubiquitous joints are:
1. slip and separation along a plane cannot be measured. Engineering judgment must be used to decide the magnitude of movement; and
2. erroneous results may occur if bedding plane properties are assumed to be the same as joint properties.
The advantages of using interfaces are:
1. slip and separation along a plane is measurable; and
2. bedding plane response in high horizontal stress fields are more accurately represented.

Likewise, the disadvantages of using interfaces are:
1. the model geometry would need extensive configuration if a lot of interfaces are required; and
2. the interface properties are difficult to determine.

It is envisaged that the WRS1 model will trial discontinuities represented by ubiquitous joints and interfaces.

**Constitutive model**

In order to reduce the runtimes and eliminate the lack of plotting features in modified constitutive models, it would be advantageous to use a built-in constitutive model that can represent strain softening and can incorporate discontinuities as either ubiquitous joints or interfaces.

If it is decided to use interfaces to represent discontinuities, the strain softening model will be used. On the other hand, if ubiquitous joints are introduced, then the bilinear strain hardening/softening ubiquitous joint model will be sufficient.

**Loading type**

From the results of initial modelling, it can be seen that loading type 2 was far from suitable for further investigation and loading type 1 produced more realistic results. However, loading type 1 assumes that lateral displacement is the same on both sides and does not vary with depth. Examination of the survey data from survey line 3 reveals that the west side of the valley closed in approximately 100 mm and the east side of the valley closed in approximately 200 mm. It is proposed that lateral displacement be applied to the model as dictated by survey measurements and be kept constant with depth.

**CONCLUDING REMARKS**

From the results of initial modelling, it is felt that the WRS1 model is capable of replicating the essential components of valley base failure, even in its simplified state.

The next stage of modelling will be aimed at refining and completing the model. This will include the selection of constitutive model, refinement of the model geometry, incorporation of discontinuities and selection of loading type. It is envisaged that once the model is completed, it will be validated with field observations and the resulting model construction guidelines will prove useful for application to other field sites.

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Practical Considerations in Longwall Support Behaviour and Ground Response

T P Medhurst

ABSTRACT

This paper examines the interplay between longwall support design/geometry features, operational controls and geological features on ground response. Expert periods of several investigations is used to demonstrate the influence of factors such as longwall support capacity and geometry, setting pressure, coal seam strength and stiffness, tip-to-face distance and hydraulic supply and control system parameters on longwall ground response. These factors are then used to outline the requirements on key controls such as retreat rate and cutting height and their influence on allowable roof convergence.

The ground response curve concept as a means to provide graphical representation of longwall support and strata interaction processes is presented. The approach was developed to address the requirement for a practical longwall support evaluation and selection tool that can take account of support load influences such as changes in roof geology or cover depth. An example of the comparison between single pass longwall and top coal caving in a thick seam environment is given to demonstrate the influence of the various factors discussed.

GROUND CONTROL AND RISK

Poor ground response and the associated business impact of longwall downtime is a major issue for the Australian underground coal industry. Geological features such as thick overlying sandstone channels, very weak immediate roof conditions, high stresses and highly cleated and friable coal seams are common. A more detailed understanding of longwall support and strata interaction processes is needed. Such an understanding requires a multi-disciplinary approach taking account of the mechanical, structural and geotechnical influences on longwall support performance.

On close examination of both mining and civil tunnelling industries, geotechnical risks can be virtually eliminated when a suitable monitoring and operational support program is implemented. The key feature being that the level and detail of the monitoring and support program must match the project risk profile. For example, significant fall-of-ground incidents are rare in the tunnelling industry. This can also be said of gateroad development operations, where optimisation of mining and development of appropriate action plans. Such processes are well developed for assessing and managing roadway stability and comprise a core role of the site geotechnical engineer. Given its success in development operations and gateroad stability, there is a unique GRC. Optimal ground support is achieved by allowing the available strength of the rock mass to be used whilst minimising the loads taken by the ground support. This is the basis of convergence monitoring, establishment of trigger levels and remedial support plans that form part of the strata management plan. A typical relationship between roof convergence trigger levels and the GRC is shown in Figure 2.

For each set of conditions (changing geology, cover depth or stress levels) there is a unique GRC. Optimal ground support practices rely upon monitoring of ground behaviour and development of appropriate action plans. Such practices are well developed for assessing and managing roadway stability and comprise a core role of the site geotechnical engineer. Given its success in development operations and gateroad stability, there is significant scope to reduce geotechnical risk through application of these principles to longwall ground behaviour. The GRC provides a convenient means to show ground behaviour, its relationship to shield performance, and roof stability within the broader context of longwall operations.

GROUND RESPONSE CURVES

Strata management plans with trigger levels based on measured convergence would be familiar to most in the coal industry. Such an approach has been used for many years and was originally developed for the civil tunnelling industry. In geotechnical engineering it is known as the observational method, in which the timing and method of ground support is determined via support pressure and convergence monitoring during construction. The underlying tool of the observational approach is the ground response curve (GRC). The general concept is outlined by Brown et al (1983) and shown in Figure 1.

The GRC shows the relationship between roof convergence and the support pressure applied. Upon excavation, initial roof relaxation occurs which would require the support resistance to match the primary stress level to prevent any convergence (Point A). As the roof begins to deform, the required support resistance to prevent further convergence reduces, as arching and the self-supporting capacity of the roof becomes utilised (Point B). The roof then reaches a point where failure begins to develop (Point C). Required support resistance then begins to increase as self-supporting capacity is lost, and support of failed ground is required (Point D).

The ground support line (PB) shows a typical point at which ground support might be installed following initial roof convergence (δ). The slope of the line (PB) reflects the support stiffness. The aim is to operate as close to Point C as possible provided that the corresponding roof convergence is tolerable, thus allowing the available strength of the rock mass to be utilised whilst minimising the loads taken by the ground support elements. It is also possible for the support to be too stiff, or installed too early, so the load bearing capacity of the ground is not fully mobilised and the load in the supports are too high. Similarly, ground support which is too soft, or installed too late, will be ineffective in controlling roof convergence.

The roof convergence monitoring and support design philosophy outlined in Figure 1 has been applied to gateroad development and roof support design for several years on an informal basis. Typically, primary support is designed/installed and then monitoring is used to guide decisions for secondary support. This is the basis of convergence monitoring, establishment of trigger levels and remedial support plans that form part of the strata management plan. A typical relationship between roof convergence trigger levels and the GRC is shown in Figure 2.

For each set of conditions (changing geology, cover depth or stress levels) there is a unique GRC. Optimal ground support practices rely upon monitoring of ground behaviour and development of appropriate action plans. Such processes are well developed for assessing and managing roadway stability and comprise a core role of the site geotechnical engineer. Given its success in development operations and gateroad stability, there is significant scope to reduce geotechnical risk through application of these principles to longwall ground behaviour. The GRC provides a convenient means to show ground behaviour, its relationship to shield performance, and roof stability within the broader context of longwall operations.

STRATA-SUPPORT INTERACTION

Two basic models exist for analysing support loading, namely force-controlled or convergence-controlled roof behaviour (Barczak, 1990). Historically, support load was estimated assuming an overlying detached roof block to be maintained in equilibrium by the support resistance (Wilson, 1993). The premise of the detached roof block approach is force-controlled.
roof behaviour. It does not consider load development resulting from main roof convergence nor does it consider the influence of the stiffness of the total ground supporting system on face convergence.

Evidence from several longwall mines operating at greater depths and/or under massive roof conditions suggests that convergence-controlled roof behaviour is generally more applicable to support response. This approach relies on determining the load distribution between the coal seam, roof strata, longwall supports and goaf, which is a function of the relative stiffness of each supporting element. The ‘overall stiffness’ of the four main support elements governs the amount and rate of roof convergence.

To assess convergence driven roof behaviour requires the use of the GRC. A typical example might be to investigate the effects of poor hydraulics on support performance and roof convergence, as demonstrated in Figure 3. A typical range of roof conditions is shown by the upper and lower GRCs. Clearly, roof degradation over time will result in higher loads and increased convergence. The support setting line shows the point at which the roof
supports are set. An 800 t longwall support with 80 per cent set-to-yield ratio is shown. Usually an amount of initial roof convergence occurs, then as the supports are set, additional roof convergence then taken up as leg closure (due to compression of the hydraulic fluid). Depending on the self-supporting capability of the strata, roof convergence would cease at the point where the support setting line meets the ground response curve. If the support does not have sufficient capacity or is set too late, roof convergence would continue as the support goes into yield.

Hydraulic leakage effectively increases the convergence permitted between set and yield. This shows how poor hydraulic maintenance can severely limit support effectiveness and contributes to poor face conditions. The net effect of hydraulic leakage is the reduction in support stiffness and setting loads. In contrast, fully operational supports (theoretical set-to-yield profile) set under similar conditions would be expected to provide stable roof conditions.

LONGWALL SUPPORT CAPACITY

In order to perform an assessment the four main support elements about the longwall face, namely the coal seam, roof strata, longwall supports and goaf, need to be considered. There are several data sources available to estimate support parameters, particularly from an operating longwall face. In general several input sources can be used:

- monitored leg pressure values from the operating longwall face;
- leg convergence/stiffness test results usually supplied by the longwall manufacturer;
- underground observations of coal seam and face conditions and associated measurements of coal seam strength and stiffness characteristics;
- goaf geometry from subsidence data and other sources such as surface-to-seam extensometers or microseismic monitoring; and
- routine geotechnical data such as roof strength from laboratory data and/or borehole geophysics.

Using leg pressure values, leg stiffness values and underground observations a rudimentary strata-support interaction diagram can usually be derived. Provided that the longwall face is not loaded to the point that the yield valves are continually activated, the leg pressure distribution along the face can provide a range of loads that can be matched against face conditions. A measure of face conditions can be deduced from an estimate of coal seam compression, leg stiffness data and underground observations, which in turn can be matched against coal seam strength data.

This approach provides the capability to graphically represent typical longwall support response from real operating data. To augment the initial assessment, the GRC also provides a series of data points that can be used to calibrate numerical modelling analyses. Modelling may then provide the means to assess the impact of variance on the existing operating conditions. One example is the recent work carried out at Ulan Mine (Medhurst and Reed, 2005).

A series of analyses were undertaken to examine strata-support characteristics for a number of modern two-leg support systems. Due to the effect of different support types, it was more convenient to present the results in terms of load density rather than load. The resulting GRCs are shown in Figure 4. Under normal operating conditions, the analyses indicate that two-legged supports imparting a load of 100 t/m² or greater would be adequate for the future Ulan operation.

Fluctuations in hydraulic line pressure are common on longwall systems, for example if several supports are activated simultaneously or hydraulic leaks develop. Variations in setting pressure across the face can therefore often lead to uneven roof loading and roof stability problems. Figure 4 shows that the 110 t/m² supports could be set at 80 per cent or possibly even 70 per cent of yield load in order to accommodate support load variance whilst limiting excessive roof convergence. The recommended support configuration for Ulan included 2 m wide, two-legged supports with a support density in the range 100 - 110 t/m². The upper limit at yield load was suggested to provide passive resistance in the event of heavy weighting, for example at panel startup or when mining through structures.
The onset of face spall at about 3 m and 4.5 MPa. A typical 3 m cutting height over a longwall face puts seam stiffness as low as 2 GPa and mass unconfined strength of a weak, cleated coking coal seam such as the Bowen Basin might. For example, a typical thermal coal seams can be used to demonstrate the possible impact on longwall support response. For example, a typical thermal coal seam such as Ulan with a strong, dull coal might have a stiffness in the order of 10 mm to 15 mm for both the coal seam and the longwall supports. The successful application of longwall support technology in recent years therefore might be partly due to close matching with coal seam stiffness characteristics to ensure good roof control. A uniform vertical compression profile helps to minimise the effects of mining induced shear stresses in the immediate roof.

OPERATING FACTORS AFFECTING STABILITY

Canopy tip-to-face distance

Roof stability is a function of lateral confinement, which is generated by the support resistance and the coal seam. In general, stability of the roof strata is highly dependent on the span-to-thickness ratio of the roof beam. Two basic principles apply:

- rock strength must be high enough to resist failure if the beam is thin; or
- the beam must be thick enough to be able to generate lateral confinement.

Roof stability is dependent on the spanning capabilities of the individual beds within the roof unit. For typical Australian roof strata in which rock strengths (UCS) are greater than 20 MPa, it has been found that long-term stable roof generally prevails when the span-to-thickness ratio ≤ 4. In other words, for bedding spacing of about 0.2 m, a canopy tip-to-face distance up to about 0.8 m would remain stable.

In some cases when bed spacing is thin and/or bedding surfaces are weak, the immediate roof skin can often delaminate. In one example, as mining activity progressed below 200 m depth, the immediate roof coal had started to fall at irregular intervals across the face. The penny band separating the coal ply from the overlying mudstone provided a convenient delamination plane. A simple unsupported span delamination model applied.

Figure 5 shows the relationship between factor of safety (or stability) and depth for a 0.3 m thick coal roof beam. The plot shows the influence of horizontal confinement on roof stability. This can be affected by the amount face spall, which in turn, can result in the forward abutment moving further into the solid coal with loss of confinement and/or clamping stress on the roof beam. It may also be affected by the lower-advance-set cycle of the roof supports, for example, the influence of contact advance.

Hydraulic supply and control settings

The importance of reliable positive set pressure across the entire faceline has been emphasised many times in discussion on maintaining face stability. All too often the effects of faulty...
blipper valves and/or inadequate pump pressure have been known to result in adverse face conditions. In general, specification of set pressures needs to take account of factors such as:

- coal seam yield/face spall,
- extra load during support advance,
- need to minimise roof convergence, and
- extended downtime.

It is noteworthy that the impact of face spall (based on 1 m of broken coal in advance of the supports) would typically be expected to result in a minimum of 40 tonnes additional load on each support. Similarly in poor conditions, if methods such as double chocking are employed, additional 50 t weighting cycles may be imposed in the roof and adjacent supports during support advance.

In weaker conditions, more frequent and/or out-of-sequence support moves often result in higher demand on the hydraulic supply system. It is therefore important to ensure that the hydraulic supply and control settings are matched to the load demand on the face. Modern hydraulic supply control systems commonly use pressure threshold values that control when adjacent supports are to operate, triggering of positive set and reactivation threshold, and sufficient pressure to push the AFC. The basic support control parameters can be often changed from default values without recognition of their impact on support performance. The main factors that need to be considered are:

- Is sufficient supply pressure reaching the centre of the face?
- In situations of high hydraulic supply demand, supply pressure to the legs may be low. Is the threshold value sufficient to ensure positive set is activated?
- Is the differential between nominal set pressure and positive set reactivation set at the right value? In some instances, this setting can result in repeated loading or ‘pumping’ of the supports on a continuous cycle, particularly for older legs that normally have a measurable leakage rate.
- Is the pumping rate sufficient to ensure correct setting times and support advance speed? Is initial roof convergence beyond acceptable levels before adequate set pressure is achieved?

**Cutting height**

The introduction of longwall mining into thick seam environments has raised new and challenging issues in ground control. As previously mentioned face stability and the associated matching of coal seam and longwall support stiffness is critical to successful longwall mining. In general, the higher the cutting height, the greater potential for face spall and then larger canopy tip-to-face distances. Anecdotally, it is well known that reduction of cutting height can have a favourable impact on longwall face stability. In poor ground conditions, it may be therefore advantageous to have a suitable working range of the supports to temporarily lower the cutting height. Ground response curves for different depths of cover and cutting heights under typical Bowen Basin conditions is shown in Figure 6.

In terms of normal ‘static’ performance, Figure 6 illustrates how the support resistance at 4.5 m height is barely adequate at depths of 250 m but is improved by lowering the cutting height to 3.8 m. Also note that approximately 30 mm of roof convergence could be expected prior to setting the supports in the lower-advance-set cycle when operating at the greater depth. This presents a situation in which the margin for error in support operation is significantly reduced. Small amounts of additional roof convergence are likely to result in roof guttering, which can easily be exacerbated by factors such as poor set pressures or inadequate hydraulic supply issues.

A large working height range for longwall supports can offer both advantages and disadvantages. Apart from the requirements of shearer clearance during cutting and transport considerations, the supports need to be able to provide active thrust to the roof in all situations. There are two main considerations:

- the canopy tip generally moves in a vertical locus plane over the working range; and
- support geometry and leg size have been designed to ensure sufficient stiffness and stability at high working heights.

The tip-to-leg distance of modern two-leg supports is commonly about 3.7 m. At cutting heights greater than 3.7 m, the supports are therefore required to operate under conditions in which the main support zone is higher than it is wide. In essence at cutting heights greater than about 3.7 m, the supports go past the ‘square’ and
In general, the time-dependent effects on caving and support loading are not well understood. However, the background rate of roof convergence is important in controlling roof stability and can sometimes be related to the impact of a slow retreat rate. In general, most Australian longwalls would operate under typical convergence limits as follows:

- face spall initiated after 15 to 20 mm vertical compression in coal seam;
- cavity development when roof convergence exceeds 30 to 50 mm; and
- overlying strata broken when roof convergence exceeds 100 mm.

In heavy weighting environments, longwalls are routinely subjected to loads that result in a convergence rates in the order of 10 mm/h. Similarly, convergence rates during a weighting cycle can typically exceed 20 mm/h. This suggests that roof cavities will develop over a period of a shift or less under slow retreat on the basis of exceeding a critical convergence level of 100 mm.

Retreat rate, stand-up time and convergence

In the discussion of canopy tip-to-face distance, it was noted that roof stability is a function of lateral confinement, generated by the support resistance and the coal seam. This is particularly important for operating under weak immediate roof conditions. The preceding discussion outlined some issues relating to delamination of a weak immediate roof layer. Another important issue however, is shearing and cavity development in thicker roof layers, often in the presence of a rider seam or weak clay layer within the lower 2 m of the roof horizon.

The shear strength of weak rock increases significantly with confinement. In basic terms, the coal seam and the longwall support legs act as the main abutments for arching of the immediate roof strata. The canopy itself then serves to provide active pressure within the arch zone. This support mechanism however, breaks down if either abutment is lost by:

- significant amounts of face spall or initial seam compression leading to a wider arch, that is beyond the span limits of the immediate roof beam; and/or
- inadequate set pressure, which is below the active pressure requirements and allows roof convergence beyond stable limits.

This problem often manifests itself when the weak layer is between 1 m and 2 m into the roof. This is because the influence of the bearing pressure of the support canopy is diminished 1 m or more into the roof and stability becomes dependent upon the self-confining effects of the roof strata. The main control in such situations is to preserve the end constraints of the roof beam so that lateral constraint can develop. In other words, damage or face spall in the coal seam needs to be minimised. An example of the effect in a thick seam environment, in which the effect of reduced cutting height increases confinement in the immediate roof and seam zone; is shown in Figure 7.

The confining effect on roof stability is sometimes counter-intuitive as it is common to focus solely on the roof strata properties alone and its potential to delaminate. The potential for shearing, however, is a localised stress related phenomenon and to some extent can be managed by the choice of a suitable cutting geometry and complimentary longwall configuration. In more extreme situations, ground improvement methods are commonly used to consolidate the face and immediate roof strata. There are many examples of the need to inject PUR into the coal seam for a significant distance ahead of the face to ensure a self-supporting roof beam can develop.

Anecdotal evidence indicates that maintaining a critical minimum retreat rate can often mitigate the effects of poor face stability. A typical first quartile Australian longwall operation cutting at 3 m height has an average annualised retreat rate of 11 m/day over a typical longwall panel. This equates to a typical daily retreat rate of 20 m/day and a maximum of 30 m/day.

The extent of the damage to the coal seam in front of the face would typically be in proportion to the seam thickness, say 2 - 3 m for a typical longwall. Therefore to maintain relative competent ground ahead of the face, a minimum retreat rate in the order of 5 m/day is warranted. However, as the size of the damage zone grows, the effect of shearing ahead of the face becomes more pronounced. Shear failure of roof material in thicker seam environments can result in damage 5 m to 10 m ahead of the face. Study of daily retreat rates in weak seams indicate that average retreat rates in excess of 10 m/day are required to limit the influence of time-dependent face loading issues and related longwall delays. Retreat rates less than 5 m/day for two consecutive days or more often result in development of cavities and poor canopy/roof contact.

GEOTECHNICAL FACTORS AFFECTING STABILITY

Weak immediate roof

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Massive overburden strata

The presence of overlying sandstone channels presents an additional consideration in relation to periodic weighting issues. Massive strata beams commonly break at a minimum length to thickness ratio of 1:1 and can be up to 2:1, especially in cross-bedded sandstones. The strength, distribution and character of the overlying sandstone units presents several issues for longwall mining, including:

- cantilever effects that overload supports under ‘massive’ conditions (including panel start-up);
- detachment of large blocks that are able to overload supports; and
- development of small blocks in tip-to-face area that disrupt cutting.

Clearly, the closer the massive strata unit is to the seam, the greater influence on support loading that is developed, particularly when the strata is able to bridge and overhang behind the canopy. This is shown in Figure 8. The net effect of overhanging strata is that the centroid of the block moves from forward of the legs to behind the legs. The length of overhang then becomes critical for support loading. Its effect is demonstrated in Figure 9 and shows how support capacity can be rapidly exceeded as a result of strata overhang.

Thick seam

Most Australian coal operators have access to potential thick seam longwall mining reserves and are looking to maximise the return on investment in these mines. The preceding topics point out several issues that require extra consideration in the thick seam environment, namely the matching of coal seam and longwall support stiffness, support geometry, retreat rate and maintaining face stability for roof control. One key issue is the potential for the cave line moving over the support canopy.

Due to geometrical factors, it is more likely for the cave line to develop above the canopy in a thick seam operation. For two-leg supports this presents a unique situation. The canopy essentially acts as a fulcrum over the leg hinge-point; therefore the canopy tip load is dependent upon the opposite reactive load behind the line of the legs. The effect is demonstrated in Figure 10.

As the distance behind the legs reduces, support pressure in this area of the canopy increases. The pressure in the immediate roof behind the legs increases until localised crushing develops, which in turn, can result in a progressive weakening of roof over the canopy. As the cave front moves forward, canopy tip load reduces approximately in proportion to distance. The net effect is zero tip load when the cave line reaches the line of the legs and the tips begin to be pushed down.

Whilst this effect is detrimental to conventional longwall mining, the very same mechanism is exploited in the longwall top coal caving (LTCC) method. LTCC supports are of four-leg design (to eliminate the fulcrum effect) and are also of lower capacity to facilitate caving over the canopies. The performance of a typical 620 t LTCC support along with support capacities of 800 t and 1000 t for a typical 8 m thick weak Australian coal seam (3 m cutting height) at 200 m and 400 m depths is shown in Figure 11.

In the 400 m deep case, the amount of initial convergence is in the order of 80 to 90 mm. This is a key issue for face stability and would result in a large damage zone above and in front of the supports. Anecdotal evidence suggests that Chinese coal seams tend to be more ‘blocky’ than the weaker Australian coking coal seams. Blockier coals tend to produce high shear strengths and are stiffer; enough to maintain face stability whilst at the same time weak enough to cave. The analysis suggests that LTCC support capacity in the order 900 t or greater might be required in a typical Australian panel layout in deep (+300 m) conditions.
One of the most common problems encountered is when operating under weak immediate roof. Invariably, poor roof conditions force operators to turn the positive set system off to maintain a consistent canopy attitude. This in turn usually leads to poor set pressures across the face and exposes the longwall to increased roof convergence as a result of hydraulic leakage (Figure 3) and other factors such as poor canopy/roof contact. One possible solution is to reconfigure the posi-set system, which is currently based on pressure control, to include a leg convergence based control parameter. In other words, once the supports are set against the roof, the posi-set system is activated to maintain the support within an allowable convergence limit. This will require appropriate sensor technology to measure convergence, presumably either by a potentiometer system, tilt sensors or leg fluid flow sensors. Factors associated with support geometry may also need to be considered.

Accurate measurement of leg convergence may have many benefits, particularly when the longwall is often operated in yield. For longwall automation purposes, it could be linked to horizon control in the lower-advance set cycle. Similarly, the use of face monitoring data is becoming more prevalent for predicting weighting cycles and support diagnostics. The leg convergence rate reflects the work done by any given support. The on-line measured work rate of a longwall support can provide a fundamental measure of its life cycle attributes as well as to reflect load transfer effects such as heavy weighting (Crisafulli and Medhurst, 1994).
ACKNOWLEDGEMENT

The input and support provided by representatives of Xstrata Coal Australia, Anglo Coal Australia and BMA are kindly acknowledged.

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Fig 11 - Strata/LTCC support interaction in weak coal.
ABSTRACT

In the ten years since the University of NSW proposed a pillar design methodology for bord and pillar operations, the Australian coal mining industry has changed substantially. What was primarily a bord and pillar design approach is now being applied to chain pillar design in longwall mines where the requirements are substantially different. The dimensions of chain pillars can impact on tailgate conditions (roof, rib and floor), seal performance, and surface subsidence. The status of chain pillar design practice in Australia is reviewed, with a focus on defining pillar strength, chain pillar loading, and assessing the performance of the roof/pillar/system. A new pillar strength equation is proposed for Australian coal that applies for all width/height ratios. An alternative analysis of probability of failure of chain pillars is presented.

INTRODUCTION

In the order of 30 gateroads are commenced in Australian longwall operations every year, with chain pillars defined primarily by two heading systems. Like any design in geotechnical engineering, the dimensioning of these chain pillars must consider both stability and serviceability:

• acceptable overall stability as well as local stability of the structure; and
• the induced movements must be acceptable, not only for the structure being considered but neighbouring structures and services.

Specifically, chain pillars are required to provide serviceable tailgates for ventilation and secondary egress by having acceptable roof and rib conditions, to allow for the construction of adequate goaf seals, and possibly to minimise surface subsidence impacts. At the same time, there is a need to reduce pillar width so as to maximise coal extraction and minimise roadway development. All of this is required in a geotechnical environment with extensive development of rock fracture and breakage such that loading on the pillars is difficult to quantify.

There appears to be a general perception that design methods exist for chain pillars and guidelines for their use are available so that they can be readily applied. This paper argues that this is not the case, and that pragmatic decisions are required in the application of the range of methods available. In the absence of guidelines for the application of the methods (with the notable exception of tailgate roof serviceability) mine designers are often required to set the guidelines and then design against them. This process often draws the attention of regulators.

This paper is concerned with pillar design for mining practitioners and so its focus is on limit-equilibrium type approaches that are readily accessible at mine sites. Numerical methods are not considered in detail as they are considered to be specialist consultancy tools, valuable for research into sensitivities of various aspects of pillar behaviour. The overall stability of chain pillars when they are located between goafs is examined, not the pillars against solid coal. The major application of the paper is in the stability of chain pillars for subsidence control. The paper does not address local stability in the tailgate corner, nor does it consider the relationship between chain pillar dimensions and surface subsidence (Seedsman, 2004).

SELECTED REVIEW OF LITERATURE

Australia

In the 1990s, the University of NSW conducted extensive research into pillar behaviour and produced a procedure for bord and pillar mine design (Galvin, Hebblewhite and Salamon, 1999). The research resulted in an empirical pillar strength equation, with pillar loads readily calculated using tributary area concepts. The data base consisted of 17 failed cases of which one had a width/height ratio of 8:1 and the rest less than five. The initial pillar strength equations had similar forms to those created earlier in South Africa, and these were later modified to incorporate non-square geometries (Equation 1).

\[
\text{Strength} = 27.63e^{0.51w_{\theta}}h^{0.226}h^{0.110}(0.290[(w_{\phi}/5h)^{2.5}-1]+1)
\]

where:

\[w_{\theta} = w_{\min} \text{ for } w/h<3\]

\[\Theta_{0} = \frac{2w}{w_{1} + w_{2}} \text{ for } w/h>6\]

\[w_{\phi} = w_{\min} \text{ for } w/h<3\]

\[h \text{ is pillar height}\]

A plot of Equation 1 for a 25 m wide 100 m long pillar is shown in Figure 1 where it can be seen that the strength increases as the pillar height decreases. The rate of increase in strength is greater at width to height ratios in excess of 8:1.

Galvin, Hebblewhite and Salamon (1999) provide a table that relates probability of failure to factors of safety (Table 1), but specifically avoids making recommendations on what values to use for various design applications. The basis of the probability table is not presented. It will be shown later that it may simply be the application of a normal distribution to a population in which the coefficient of variation is 28 per cent. This relatively large coefficient of variation may be indicating that there are some additional unaccounted variables that influence coal pillar strength with the most obvious one being coal strength. It is noted that recent pillar research in South Africa is now separating weak coal from normal coal.

The relationship between factor of safety and probability of failure cannot be used for chain pillar design because of the substantial difference between the variance in the estimate of tributary area in a bord and pillar operation and the variance in the estimate of chain pillar loads. This will be addressed later.

ALTS (Colwell, 1998) uses a different pillar strength equation (Bieniawski, 1968 – Figure 1) and provides an integrated strength/stress/factor of safety recommendation for tailgate serviceability based on detailed back-analyses. Galvin, Hebblewhite and Salamon (1999) and Bieniawski (1968) give similar strength for width to height ratios less than about 7:1 but diverge for squatter pillars. If using ALTS, it is important that the same coal strength equation is used – there have been cases where ALTS recommendations regarding factors of safety are used with the higher coal pillar strength given by the equation of Galvin, Hebblewhite and Salamon (1999).

Seedsman (2001) suggests that the relationship between factor of safety and tailgate roof conditions that underpins ALTS may be related to the onset of tensile roof stresses in the tailgate roof if the chain pillars begin to yield. This large deformation sets up...
a rotation of roofline, an increase in the bay-length of the roof and a consequent loss of horizontal confinement. Seedsman (2004) suggests that the Bieniawski (1968) linear equation may be considered to be a yield equation and Galvin, Hebblewhite and Salamon (1999) may represent the ultimate strength.

Medhurst and Brown (1998) and Medhurst (1999) provide methods to determine coal strength based on the rank of coal in combination with brightness profile mapping. This should allow the use of computer models to probe the relationship between coal strength and empirical pillar strength but such a study has yet to be published.

**International**

There have been two revisions of an alternative pillar strength equation in South Africa since 1967 (van der Merwe, 1999, 2002). In 1997 there were 27 failed cases and in 2002, the database now consists of 54 failed cases, with width to height ratios of 0.9 - 3.8. The South African database has also been structured to distinguish between weak and ‘normal’ coal. The alternative equations are based on a different statistical method to that used by Salamon and Munro (1967) and Galvin, Hebblewhite and Salamon (1999) and are much simpler and more efficient in separating failed and unfailed cases. The new South African analyses are not accompanied by a factor of safety/probability of failure analysis. Van der Merwe (2002) also argues that squat pillar formulations are very subjective with inadequate control of selection of key parameters (the 5 and 2.5 values in the square brackets in Equation 1).

\[
\text{Strength} = 4.0 \, w^{0.81} h^{-0.76} \, (1999) \quad (2)
\]

\[
\text{Strength} = 3.5 \, w/h \, (2002) \quad (3)
\]

Equation 3 is plotted in Figure 1, where it gives a higher strength except at width/height ratios greater than ten. The 2002 formula is 22 per cent more efficient in separating failed and unfailed cases compared to the original Salamon and Munro (1967) formulation.

**TABLE 1**

<table>
<thead>
<tr>
<th>Probability that pillar stability is less than calculated</th>
<th>Normal statistic (one sided)</th>
<th>Factor of safety – bord and pillar loading (Galvin <em>et al.</em>, 1999)</th>
<th>Possible factor of safety – chain pillar double goaf loading (this paper)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1:10</td>
<td>1.64</td>
<td>1.22</td>
<td>1.56</td>
</tr>
<tr>
<td>1:20</td>
<td>1.96</td>
<td>1.3</td>
<td>1.67</td>
</tr>
<tr>
<td>1:50</td>
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<td>1:100</td>
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<td>1:10000</td>
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<td>1:100 000</td>
<td>4.41</td>
<td>1.95</td>
<td>2.50</td>
</tr>
<tr>
<td>1:1 000 000</td>
<td>4.89</td>
<td>2.11</td>
<td>2.67</td>
</tr>
</tbody>
</table>

**FIG 1** - Comparison of pillar strength relationships.
CHAIN PILLAR STABILITY ASSESSMENT IN AUSTRALIA

In Australia, the standard approach to assessing chain pillar stability is to use the equation of Galvin, Hebblewhite and Salamon (1999) for pillar strength and dividing it by some estimate of pillar loading to give a factor of safety. This general approach is appropriate for assessing the overall stability of the pillar system, but may not necessarily address local stability underground and does not address deformations. The assessment of tailgate roof stability is a notable exception.

Pillar strength

Current practice is to use the equation of Galvin, Hebblewhite and Salamon (1999) for effective width and pillar strength (Equation 1). Since the origin of this equation is based on earlier work in South Africa, the more recent methods of van der Merwe (1999, 2002) have been applied to the Australian database using the effective width conversion as used by Galvin, Hebblewhite and Salamon (1999). The resulting relationship (Equation 4) is 26 per cent more efficient in differentiation between failed and unfailed cases. Given the similarities in the Australian and South African databases, this level of improvement (which is similar to van der Merwe, 2002) is not unexpected.

\[ \text{Strength} = 13.52 \, w^{0.65} \, h^{1.35} \]  

(4)

Equation 4 is also plotted in Figure 1, where it can be seen that it is similar to van der Merwe (2002) for width to height ratios of less than seven, and diverges as it extrapolates from the database. The coefficient of variation in this relationship is approximately 34 per cent.

The simplicity of the empirical methods to estimate pillar strength is challenged when cases outside the database are involved. The need to extrapolate beyond 5:1 width to height, and especially 8:1 has been identified by many workers. The increased extraction of thick coal seams is introducing a set of additional variances in the loading factors.

Pillar load

Tributary area concepts can be used for bord and pillar layouts. The variance in this estimate of pillar load is very low, being related to the geometry of the pillars and the seam. The same loading cannot be used for chain pillars because of the goafing that develops in the wide unsupported spans. Methods such as ALTS (Colwell, 1999) use an abutment angle model, which proposes that the increase in pillar load can be related to the dead weight of a wedge of rock located over the side of the goaf. The values of the abutment angle quoted by Colwell have been measured from maingate and tailgate corner loading, and not from the loading of pillars with goaf on both sides. Colwell attributes the wide range of angles for maingate loading in part to possible arching and loading of the solids. Only four values are provided for the tailgate loading conditions and for these the average is 21° and the standard deviation is 3.8° – a coefficient of variation of 18 per cent.

In the absence of double goaf loading, the tailgate values are often used. For double goaf loading, consideration of arching of loads onto solid unmined coal is not required.

Factor of safety

The factor of safety is simply the quotient of the estimated pillar strength and the estimated pillar load. In terms of pillar design, the fundamental issue is the acceptability criterion applied to the quotient, and this requires the consideration of the probability and consequences of any failure of the pillar system.

The following is a simplistic examination of the relationship between factor of safety and probability of failure for chain pillars – it is certainly not statistically rigorous but is provided as a basis for discussion and for further work. To be done rigorously will require consideration of the variance of a number of ratios of parameters each of which have their own variances.

The basis of the probabilities in the study by Galvin, Hebblewhite and Salamon (1999) is not stated. If we plot the relationship between their factors of safety and failure probability by converting the probability to the normal statistic for a one sided distribution we get a straight line with a gradient of 0.28 (Figure 3). If we assume that variance in load estimate in the pillar database is small compared to the variance in the pillar strength, the inference is that the pillar strength equation has a coefficient of variation (standard deviation/mean) of 28 per cent.
Roof and floor failure

The empirical pillar strength approach to assessing pillar stability is only valid if the pillar is the weakest component of the roof/pillar/floor system. Papers on empirical design correctly make statements that the empirical approach is only valid when the roof and floor are ‘competent’ – but there is no definition of what is meant by competent. Gale (1999) reports the results of computer modelling with different roof and floor conditions but once again strong and weak are not defined. In Figure 4, the relationship between average pillar strength and width/height ratio is shown together with the four lines from Figure 1. The fact that the empirical relationships lie between strong and weak is very encouraging and suggests not only that the extrapolations beyond 8:1 width to height ratios are valid but also that implicit in the database are strong and weak roof and floors.

From a practical viewpoint, the inability to generate confinement in a coal pillar will relate to the onset of slip on low shear strength layers in the roof or floor (the so-called slippery layers) or if there is failure of the roof or floor mass. The former case may only be found when the friction angle of the surfaces are less than say 10° – geologically this requires bedding plane thrusting along planar surfaces. This is possible, but it is unlikely to be encountered due to the incompatibility of such geological conditions with high production longwalls.

A particularly important case is the possibility of the failure of low strength floors at stresses less than those that would cause failure of the coal itself. Such floors can be encountered in some areas of the Great Northern Seam in New South Wales and at shallow depths under the Tertiary unconformity in the Bowen Basin. Bearing capacity assessments, as used in foundation engineering, can be used with a need to use comparatively high values of factor of safety (Li et al, 2003). Being basically elastic methods, they are scale independent and so can be used for pillars – the high factors of safety are required given the lack of use of this application (greater use will enable more rational values).

CONCLUSIONS

For chain pillar design, there are a number of tools available to the mining engineer that are based on empirical approaches or well-established analytical methods as used particularly in soil mechanics. In common with all geotechnical engineering practice, the tools should be used in a design process that includes data gathering prior to analysis, back analysis/calibration against early layouts, and observation and monitoring.

Whilst there is no statistical validity for the empirical pillar strength method for width/height ratios in excess of eight, and possibly in excess of five, the agreement with computer analyses is encouraging. The currently used empirical pillar strength equation may be underestimating pillar strength by about 20 per cent. There is a need to consider the definition of pillar height in thick seams.

The factor of safety/probability of failure relationship for bord and pillars does not apply to chain pillars. More work is required to determine the variance in the estimate of chain pillar loading.

The major obstacle to chain pillar design is the lack of an agreed acceptability criterion. In the meantime, the pragmatic way forward is to find similar and acceptable mining layouts and use them to provide a local ‘calibration’ of the design tools – this will result in the continuation of conservative designs. A more sophisticated statistical analysis is required. The focus needs to shift from pillar strength research to implementing a design.

REFERENCES


Loading Mechanics of the ‘Can’ and Implications for Improved Strength and Stiffness Properties

G Tarrant¹

ABSTRACT
Improved roof control in high deformation tailgate environments has been achieved over the last decade through development of stiffer and increased capacity standing support products. ‘The Can’ is one such development, being a steel cylinder containing a weak cementitious fill, designed to essentially fold in upon itself whilst maintaining strength. A laboratory study to better define the relative load contributions of the steel cylinder and fill, the confining interaction between these components and most importantly, the potential impact of varying the steel and/or fill properties is described. This would enable the support engineer to ‘dial-up’ the desired strength and stiffness properties and optimise the standing support design with respect to load capacity, stiffness, weight (handling) and cost. Design curves to optimise strength based on steel casing thickness, fill strength (confined and unconfined) and ‘Can’ geometry were established. Scaled-down (one-third) samples were used in the test program and found to adequately reflect the loading behaviour of full-scale versions, thereby providing significantly greater scope for further product development at less expense compared with testing full-scale products.

INTRODUCTION
A laboratory study of the interaction between the steel casing and fill material of ‘The Can’ standing support is described. Whilst the overall load/deformation characteristics of the product had previously been obtained by full-scale laboratory tests conducted by NIOSH (Pioneer Burrell, 1995), the relative contribution of the steel cylinder versus filler had not been established. This information was required for the development of stiffer and higher capacity (or softer and lower strength) systems. Unfortunately, the cost and logistics of conducting full-scale tests were prohibitive so the study was conducted using scaled-down (one-third) versions of the product.

The objectives of the testing program were:

• to measure the confinement provided by the steel casing to the filler and the consequential increase in strength of the fill material;

• to relate the scaled-down tests to full-scale versions and thereby establish the applicability of using scaled-down versions in product development;

• to establish design criteria regarding steel thickness, filler properties and ‘Can’ geometry; and

• to better understand the field loading behaviour of the product.

Three mini-cans were tested, one of which was tested as an empty steel cylinder. Each can was instrumented with 20 strain gauges to measure axial and circumferential strains. The study objectives were achieved, thereby providing the support design engineer with the ability to ‘dial-up’ the desired support properties of stiffness and strength within practical limitations such as weight and cost.

BACKGROUND

The product
‘The Can’ was developed by Burrell Mining for in the USA on a ‘yieldable confined core concept’ (UPSTO, 1994). It is composed of a cold rolled steel cylinder filled with a foamed cementitious blend including flyash. The ‘Can’ is typically handled underground using an Eimco with claw attachment which currently constrains the weight of the product to approximately 2.0 t. Its use is widespread throughout the coal mining industry.

Generally if a higher capacity product is desired, then a larger diameter ‘Can’ is used. However handling limitations and other aspects such as the disruption to ventilation and access are also important issues that limit the strength achievable. A 915 mm diameter ‘Can’ is typically the largest used with a yield of approximately 160 t. In the absence of further testing, the opportunity for the mine engineer to optimise support cost against other variables such as support capacity, density, size or handling is limited.

Figure 1 illustrates characteristic load/deformation profiles of various standing support products, including the ‘Link n Lock’ and pumpable cement systems. The strength of the largest products in widespread use is limited to approximately 160 t for the high yield types. Greater capacity is achievable through the pumpable products however there is a rapid reduction in post yield strength for these types. It is emphasised that the purpose of this paper is not to discuss the benefits of one product over another since they all have application in differing environments.

![Figure 1 - Comparative load/displacement of selected standing supports (after Barczak, 2000).](image-url)
Previous research

Concrete filled steel tubes (CFTs) are used within civil construction due in part to the economic benefits of using concrete versus steel. Substantial research into this area has been conducted over the last 60 years (O'Shea and Bridge, 1994, 1997a, 1997b; O'Shea, 1998; Morino and Tsuda, 2002) with the initial focus on thick walled steel cylinders and normal strength concrete (15 to 50 MPa). The increased cost of steel has driven research towards thinner steel tubes and use of higher strength concretes (100 MPa). Aziz et al. (2001) conducted a range of tests on 150 mm diameter, 500 mm high steel tubes (1 mm thick) filled with a variety of low strength fillers between 3.6 and 22.9 MPa. The work found that changes in filler strength influenced the bearing capacity of the composite columns. The body of existing research provided some insights into the behaviour of CFTs in general and particularly in relation to:

- the strength of the steel tube component,
- the possible load distribution between concrete and steel, and
- the ductility of the steel/concrete composite.

Essentially the concrete core interacts with the steel casing only after yield of the concrete occurs. The resistance provided by the steel casing may increase the post yield strength of the concrete depending on the relative strain characteristics of the concrete and steel.

The total load developed by a CFT can be separated into the contributions of the concrete, the bare steel tube and confining effects of the steel provided to the concrete. According to the research, the maximum contribution of the steel cylinder can be determined independently of the fill using existing buckling formulae (AS 4100, 1990; Grimault and Janss, 1977; AISC-LRFD, 1994). The presence of the fill doesn’t enhance the load at which buckling occurs since buckling is usually directed outward, not inwards. The contribution of the concrete can be separated into its unconfined and confined components.

The post yield behaviour of a CFT and its ultimate strength depend on whether or not the CFT exhibits strain hardening or softening characteristics. This is again a function of the radial strain characteristics of the concrete and the confining response of the steel casing.

Composite strength of concrete filled steel tube

Figure 2 illustrates the stress conditions in the steel tube and concrete core. From the equilibrium of forces a relationship between the hoop tensile stress $\sigma_h$ and the internal pressure $\sigma_p$ can be established (Equation 1).

$$\sigma_p = \frac{r}{t} \sigma_h$$

where:

- $r$ and $t$ are the radius and thickness of the steel respectively

The strength of the confined concrete is given by the following relation:

$$\sigma_{tc} = UC_{S} + TSF \cdot \frac{r}{t} \sigma_h$$

where:

- TSF is the triaxial stress factor given by $(1+\sin\phi)/(1-\sin\phi)$
- $\phi$ is the internal angle of friction
- $\sigma_{tc}$ is the strength of the concrete
- $UC_{S}$ is the unconfined compressive strength
- $\sigma_{3c}$ is the lateral confining stress

The total load in the CFT can now be written in terms of the sum of the contributions from the concrete and steel according to Equation 4.

$$L_T = L_C + L_S = \left(UC_{S} + TSF \cdot \frac{r}{t} \sigma_h\right)A_C + \sigma_{3c}A_S$$

where:

- $A_C$ and $A_S$ are the respective areas of the concrete column and steel (note that an effective area approach would be used for the load in the steel)
- $L_T$ is the total load
- $L_C$ is the load in the concrete
- $L_S$ is the load in the steel
Equation 4 provides a relationship between the components of the composite system. The fill properties of UCS and TSF can be determined by triaxial testing in the laboratory and the load at yield of the steel cylinder can be estimated from empirical formulae or also determined through testing. The radial strain of the concrete and the consequential resisting confinement developed by the steel casing is obtained either empirically (as in this study) or numerically through use of FEM code.

**Implications for mini-can tests**

The measurement of hoop stresses in ‘Cans’ has never been conducted and following a request for information from the manufacturer, the buckle strength of empty ‘Cans’ has also never been conducted. No published data exists for the triaxial properties of the fill typically used in the ‘Can’. If development of the ‘Can’ was to include analytical evaluation (use of Equation 4) of different fill types, steel cylinder geometries or casing materials, the following aspects required either measurement or confirmation:

- The applicability of the empirical equations developed for bare steel tubes for the very high D/t ratios should be evaluated. The mining application of CFTs are characterised by very high D/t ratios (>450) compared with the civil application (D/t ratios typically <90).
- The triaxial strength properties of the fill required determination. The fill typically used in the mining application of CFTs is very weak (<3 MPa) compared with the civil application (15 to 100 MPa).
- The radial strain behaviour of the fill and the confining response of the steel tube required measurement.

The mini-can laboratory tests were designed to gain a better understanding of these aspects.

**TEST SERIES**

**Test specimens**

The mini-cans were manufactured by Pioneer Burrell without specific instruction regarding the welds or degree of roundness. Each mini-can was constructed with a single longitudinal weld and was capped at one end. In the vicinity of the longitudinal welds, local warping of the cans was visible. The dimensions and weight of the mini-cans are provided in Table 1. One of the filled cans was 10 kg lighter, indicating the presence of a large air pocket within the specimen. Greater control over the filling procedure and checking of the weight at the time of specimen filling would be required for future tests. The filled specimens were noted to have some steel (<5 mm) proud of the fill due to uneven settlement. Due to the size and weight of the specimens, no attempt was made to machine the ends parallel. Instead, the specimens were topped with normal cement and levelled.

Given the extent of the imperfections, the small scale tests should be considered a ‘first pass’ evaluation. Future use of small scale specimens would require much tighter tolerances and development of standard procedures to reduce the variables introduced into the testing process. The effect of imperfections would be expected to become more pronounced with further reductions in specimen size.

**Fill testing**

Small fill samples of approximately 50 mm diameter × 100 mm length were poured to establish the unconfined compressive strength (UCS), Poisson’s ratio, Young’s modulus and triaxial strength characteristics. The tests were conducted by Strata Testing Services Pty Ltd according to Australian Standard AS 4133.4.3 (1993) and in the case of the triaxial tests, ISRM-suggested methods. The fill test results are summarised in Table 2. Figure 3 indicates that the strength increase versus confinement or triaxial stress factor (TSF) was approximately 1.9, which is considerably lower than the value of 4.1 typically used in the prediction of CFT strength (Morino and Tsuda, 2002) for example. The relatively low value of TSF indicates that the strength of the fill material is not enhanced to the same extent (50 per cent) as that of normal strength concrete.

**Test results**

The tests were conducted at the University of Sydney, Civil Engineering laboratory using the Dartek 200 tonne capacity machine. All tests were concentric, axial loading conducted under stroke control at a rate of 5 mm/minute. The post yield characteristics of one of the tests (number 3) were investigated under stroke control of 25 mm/minute. Ten pairs of axial and circumferential linear strain gauges were attached to each can in the configuration shown in Figure 4. The strain gauges were logged automatically by a Datafaker. The Appendix illustrates the typical strain gauge output from the tests.

**Bare steel tube**

The load versus axial shortening of the bare steel tube is shown in Figure 5. An initial seating-in of the bare steel tube was evident from the initial portion of the stress/strain curve, occurring over approximately 2 mm. The maximum load was 270 kN and yield occurred due to local buckling as shown in Figure 6.

<table>
<thead>
<tr>
<th>Specimen</th>
<th>Diameter (mm)</th>
<th>Thickness (mm)</th>
<th>Length (mm)</th>
<th>Weight (kg)</th>
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<tr>
<td>1</td>
<td>311.0</td>
<td>1.9</td>
<td>1006</td>
<td>13.4</td>
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<tr>
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<td>1.9</td>
<td>992</td>
<td>65.8</td>
</tr>
<tr>
<td>3</td>
<td>311.9</td>
<td>1.9</td>
<td>993</td>
<td>76.2</td>
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**TABLE 2**

**Fill test results.**

<table>
<thead>
<tr>
<th>Sample</th>
<th>Sample diameter (mm)</th>
<th>Sample length (mm)</th>
<th>Moisture content (%)</th>
<th>Density (g/cc)</th>
<th>Confining stress (MPa)</th>
<th>UCS (MPa)</th>
<th>E (GPa)</th>
<th>Poisson’s ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>50.8</td>
<td>110.4</td>
<td>16.3</td>
<td>0.635</td>
<td>0</td>
<td>2.4</td>
<td>1.6</td>
<td>0.21</td>
</tr>
<tr>
<td>2</td>
<td>50.9</td>
<td>109.8</td>
<td>15.2</td>
<td>0.614</td>
<td>0</td>
<td>1.9</td>
<td>1.5</td>
<td>0.23</td>
</tr>
<tr>
<td>3†</td>
<td>51.0</td>
<td>102.8</td>
<td>16.2</td>
<td>0.616</td>
<td>0.2</td>
<td>1.7†</td>
<td>n/a</td>
<td>n/a</td>
</tr>
<tr>
<td>4†</td>
<td>50.9</td>
<td>98.8</td>
<td>13.3</td>
<td>0.596</td>
<td>0.4</td>
<td>1.7†</td>
<td>n/a</td>
<td>n/a</td>
</tr>
<tr>
<td>5†</td>
<td>50.8</td>
<td>100.7</td>
<td>12.3</td>
<td>0.599</td>
<td>0.8</td>
<td>1.7†</td>
<td>n/a</td>
<td>n/a</td>
</tr>
</tbody>
</table>

† Inferred from extrapolation of triaxial test series.
Various codes have been developed to predict the buckling loads of bare steel cylinders, including Australian Standard AS 4100 (1990), AISC-LRFD (1994) and Grimault and Janss (1977). Figure 7 illustrates the range in predicted strength for these codes for 1.9 mm thick steel tubes and Table 3 is a summary for the mini-can and larger versions typically used underground.

Clearly there is a variation in the predicted loads with the AISC-LRFD (1994) code predicting significantly higher tube loads at larger diameters. This may reflect the level of conservatism applied by the respective standards committees as well as the range of cylinder geometries forming the empirical basis of the codes. O’Shea and Bridge (1997a) note that AISC-LRFD (1994) give accurate predictions except for the thinnest tubes and were also developed for cold formed steels. If it is assumed that the LRFD code would be inappropriate for this study, the remaining AS 4100 (1990) and Grimault and Janss (1977) codes provide a guide to the expected strength of the bare steel tubes. Further investigations would benefit from measurement of the bare steel tube strengths for the larger scale ‘Cans’.

The discrepancy between the tested peak load and the predicted peak load from the codes is partially attributed to the conservatism embodied in the code guidelines and partially due to the imperfections in the test specimen, particularly the uneven initial loading due to the out of squareness between the top and base of the tube. The use of smaller steel tubes would allow better tolerances for sample preparation since machining of the ends within a standard lathe would be possible.

**Filled mini-cans**

The load versus axial strain of the filled cylinders is provided in Figure 8, which also shows the bare tube test for comparison. The load at yield of the mini-cans was 46 and 48 t for samples 2 and 3 respectively. Both mini-cans yielded due to local buckling...
at the top of the cylinder as shown in Figure 9. The similar yield load for both mini-cans was somewhat surprising given that specimen 2 was 10 kg lighter than specimen 3. This aspect is elaborated further in the discussion section.

The post yield behaviour of specimen 3 was examined in greater detail. The stroke rate was increased to 25 mm/minute and continued until the stroke limit of the testing machine (225 mm). Figure 9 illustrates the development of local buckles at various positions along the can. The mini-can was noted to develop a single outward buckle at the top of the can at yield, then followed by a further buckle approximately 50 mm below the first. The mini-can proceeded to concertina until both buckles made contact.

The load versus displacement plot including post yield is illustrated in Figure 10. The peaks and troughs of the load history were noted to coincide with the concertina process of buckle development followed by closure as buckles made contact with other buckles. The load difference between peaks and troughs was approximately 10 t or 20 per cent of the load at yield. Each trough and peak load was successively higher than the previous, indicating a strain hardening behaviour.

Figure 11 illustrates the axial versus circumferential strain for gauges nine and ten, Can number 3 (refer Figure 4). The plot clearly indicates that at the onset of buckling at the top of the mini-can, partial elastic unloading of the steel occurred in both the axial and circumferential directions. Since the steel is behaving plastically only locally, once the vertical displacement has increased by a buckle wavelength, elastic reloading of the steel tube can occur.

**TABLE 3**

<table>
<thead>
<tr>
<th></th>
<th></th>
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<th></th>
<th></th>
</tr>
</thead>
<tbody>
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<td>300</td>
<td>270</td>
<td>326</td>
<td>394</td>
<td>409</td>
</tr>
<tr>
<td>630</td>
<td>Not tested</td>
<td>415</td>
<td>585</td>
<td>537</td>
</tr>
<tr>
<td>800</td>
<td>Not tested</td>
<td>468</td>
<td>721</td>
<td>590</td>
</tr>
<tr>
<td>900</td>
<td>Not tested</td>
<td>450</td>
<td>800</td>
<td>614</td>
</tr>
</tbody>
</table>

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Confinement generation

The confining stress provided to the fill by the steel casing was calculated from the averaged circumferential strains and the steel properties. An average circumferential strain of 60 µS was attributed to the interaction between the fill and the steel cylinder. The total circumferential strain in the steel cylinder is the sum of the circumferential strain associated with axial loading (Poisson effect) and the circumferential strain due to expansion of the fill:

\[ \varepsilon = \varepsilon_{\text{c,ax}} + \varepsilon_{\text{c,exp}} \]  

where:
\[ \varepsilon_{\text{c,ax}} \] is the circumferential strain due to axial loading of the cylinder
\[ \varepsilon_{\text{c,exp}} \] is the circumferential strain due to the concrete expansion

and since:

\[ \varepsilon_{\text{c,ax}} = -\nu \varepsilon_{\text{ax}} \]  

where:
\( \nu \) is the Poisson’s ratio of the steel
\( \varepsilon_{\text{ax}} \) is the axial strain in the steel

The incremental confining pressure provided to the fill after Equation 1 and recast in terms of circumferential strain is given by:

\[ \Delta p = \Delta \varepsilon_{\text{c,exp}} E_c \left( \frac{t}{r} \right) \]  

The total confining pressure provided by the steel casing to the fill at yield was calculated to fall within the range 0.15 to 0.2 MPa. The confining stresses enhanced the mini-can strength by only 20 kN out of a load at yield of 480 kN.

DISCUSSION

Deformation style

The deformation of the mini-cans was consistent with that observed underground for full-scale products, in particular the key feature of local buckle development and concertina behaviour of the steel tube. The behaviour of the mini-can was also consistent with experimental test work conducted by research into the application of concrete filled tubes (CFTs) to the civil industry. The civil research indicated that the shape of the steel section has a significant effect on the development of local buckles with circular sections generally buckling outward and square or rectangular section buckling both inward and outward. The yield strength of sections that buckle outward would not be expected to benefit from the presence of fill.

The mini-cans tests provided further insight into the post yield behaviour of the steel section. Whereas a bare steel tube will continue to deform until losing all strength, the presence of the fill results in the controlled concertina effect. In the early stages of yielding, a single buckle forms. With progressive axial displacement equal to the buckle wavelength, the steel cylinder has effectively regained its original shape and the load increases until another buckle commences. The cycle results in an oscillation of the peak load as deformation progresses.

Comparison with full-scale tests

Figure 12 illustrates the peak load achieved in the filled mini-can tests compared with the published data available for 630, 800 and 900 mm ‘Cans’ (Pioneer Burrell, 1995) and predictions of the peak load using Equation 4 based on:

- bare steel tube strength,
- unconfined fill strength, and
- confined fill strength based on measured hoop stress.

The results are summarised in Table 4. The range of predicted loads shown in Figure 12 was established using both the AS 4100 (1990) and Grimault and Janss (1977) codes for predicted strengths for bare steel tubes. The predicted ‘Can’ strengths using these two codes (for the bare steel tube strength component) were found to bracket the measured loads based on these codes.

Figure 13 is a bar graph of the relative load contributions of the steel cylinder, unconfined fill and confined fill. The basis of the graph is the predicted strength of the steel cylinders assuming AS 4100 (1990) code for bare steel tubes, the measured UCS of the fill (1.7 MPa) and the predicted confining stresses extrapolated from the mini-can tests to the larger diameter ‘Cans’. The plot is presented as cumulative to more easily identify the contribution of the confinement generated. The estimated confining stress generated for a 915 mm ‘Can’ was approximately 0.05 MPa (increasing the fill carrying capacity by 60 kN).
Implications for product development of the ‘Can’

Based on the results obtained, the role of the steel tube is to principally develop axial load and to physically restrict the movement of the fill so as not to allow sloughing however minimal enhancement of the fill strength is provided by the steel. The steel tube accounts for approximately 35 per cent of the total load in a 900 mm diameter ‘Can’.

Cuttability

A cuttable and presumably softer material would be expected to generate less confining stress to the fill compared with steel. However, given the low contribution of the confined fill (four per cent) the consequential reduction in strength from loss of this confinement would be minimal. Since the steel tube provides approximately 35 per cent of the total load for a 915 mm ‘Can’, an equivalent strength of non-steel material would be required to maintain the same load capacity. Since the steel contribution increases to nearly 50 per cent for a 600 mm ‘Can’, the requirement for the non-steel material to provide an equivalent load capability compared with the steel increases. In other words, the penalty in peak load terms for a 600 mm cuttable ‘Can’ is more severe compared with a 900 mm cuttable ‘Can’.

Steel thickness

Figure 14 is a plot of predicted ‘Can’ strength versus steel thickness for 600 mm and 915 mm diameter ‘Cans’ assuming:

- the same fill properties as that used currently,
- same steel strength properties, and

A linear increase in strength of approximately 45 or 50 t would be expected for every 1 mm increase in steel thickness for 600 mm and 900 mm diameter ‘Cans’ respectively. In relative terms, the impact of increasing steel thickness is greater for 600 mm versus 900 mm ‘Can’. This is a consequence of the greater relative contribution of the steel cylinder to the total load for smaller diameter ‘Cans’. According to Figure 14, a steel thickness of 2.5 mm for a 600 mm ‘Can’ would achieve approximately the same strength as that for a 1.9 mm casing for a 915 mm ‘Can’. As a precautionary note, the possibility of column buckling and the impact of lateral displacements should be considered in a field application of this finding.

Fill strength

Figure 15 illustrates the expected increase in (915 mm) ‘Can’ strength for an incremental increase in fill UCS for various steel thicknesses. For every 1 MPa increase in fill UCS, the ‘Can’ capacity would be expected to increase by approximately 70 t, irrespective of the steel thickness. Increasing the fill strength by approximately 1 MPa would therefore be expected to have a resulting increase in total ‘Can’ strength similar to that indicated by increasing the steel thickness by 0.5 mm. The practical drawback to increasing fill strength is the consequential increase in fill density which may increase the total ‘Can’ weight to an unacceptable level for handling underground.

Confined fill properties

The contribution of the confined fill to the total ‘Can’ strength is influenced by both:

- the confining stress generated through interaction with the steel; and
- the responsiveness of the fill to that confinement (triaxial stress factor (TSF)).

Both of these characteristics are associated with the fill properties. From Equation 2, the increase in strength for a given confinement is related to the internal angle of friction of the fill. A

### Table 4

<table>
<thead>
<tr>
<th>Diameter (mm)</th>
<th>Predicted (AS 4100)</th>
<th>Predicted (Grimault and Janss) (kN)</th>
<th>Tested (kN)</th>
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<tr>
<td>300</td>
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</tr>
<tr>
<td>900</td>
<td>1593</td>
<td>1757</td>
<td>1645</td>
</tr>
</tbody>
</table>
Recommendations to increase the internal angle of friction of the fill are beyond the scope of these investigations. In general terms the internal friction angle would be expected to increase through means such as increasing the level of interlocking between particles. This may include the relative proportions of the constituent mix, mechanical properties of the constituents, void ratio, etc.

The existing fill is like a foam with a skeletal fabric and large void ratio. The deformation of the fill is characterised by a progressive collapse of the skeletal fabric. Eventually the void ratio would be expected to decrease to the extent that the fill behaved more like an aggregate where volume increase would be expected to accompany continued deformation. Figure 9 illustrates that even after axial shortening of 22 per cent, the volume continued to decrease with increasing axial shortening (no barrelling of the cylinder evident). It is suggested that decreasing the void ratio would be expected to have the greatest impact on the level of confining stress generated by the steel and also the responsiveness of the fill to that confinement (internal friction angle). Increasing the steel thickness or limiting barrelling of the ‘Can’ by application of ring stiffeners or other means is unlikely to result in a significant increase in the confined strength of the existing fill.

**Casing strength**

In civil applications where the concrete is intended to carry the bulk of the load and the role of the steel casing is to provide confinement, disconnection between the steel and the loading system (load concrete only) has been found to be desirable under some circumstances. In the mining application, the steel cylinder was found to form a significant contribution (>30 per cent) to the total system load, even after buckling. Future product development that may result in disconnection of the steel cylinder from the roof and floor or features that reduce the load at which buckling occurs should be avoided. The introduction of longitudinal ribs or other methods to delay the onset of local buckling would be expected to increase the strength of the steel cylinder and therefore the overall system strength.

The key point is that the role of the steel cylinder in the mining application is different to the civil application. In the mining application the role of the steel cylinder is to contribute to overall strength whereas the civil application the role is more one of confinement.

**Implications for field behaviour of the ‘Can’**

**Stiffness**

The axial shortening of the mini-can at yield was approximately 7500 µs or 7.5 mm for the 1000 mm test specimens both filled and unfilled. If this result is extrapolated to an underground application for a ‘Can’ length of 3 m for example, then yield would be expected at a minimum roof to floor convergence of 22.5 mm. This value does not include seating-in of packing materials or closure of gaps between the roof and support. The displacement at which yield occurs is also independent of ‘Can’ diameter assuming that column buckling does not occur. The value of 22.5 mm would also provide a useful indicator for the required expansion of a hydraulic packer or other device if pre-stressing of the ‘Can’ were considered as a design option.

The identical value of axial shortening to reach yield for an unfilled versus filled ‘Can’ indicates that yield of the composite is controlled by yield of the steel, not the fill. This is opposite to the civil application where higher strength concrete typically results in yield of the steel cylinder first. In the underground application, the observation of a local buckle would signify yield of the system (approximately 160 t for a 915 mm diameter ‘Can’). No visible sign of system load would be provided by the...
steel cylinder prior to yield. Loading could only be implied through compaction of the packing materials.

Whilst the ‘Can’ exhibits limited strain hardening (gains strength with convergence), the presence of local buckling would signify that additional resistance against further roof to floor convergence would be minimal. In many respects the ability to ‘follow the roof down’ and at least maintain load is a deliberate design aspect of the ‘Can’ and of standing supports in general. Improvement of the strain hardening characteristics of the product would be expected to have a positive impact on roof control in high deformation areas.

**Strength**

The mini-can test program has highlighted the greater contribution of the fill (>60 per cent for 915 mm diameter versions) versus the steel to the overall strength of the ‘Can’. Since the fill is considered to be a very weak material (UCS 1.7 MPa) and has very low responsiveness to confinement, development of improved fill strength both pre and post yield appears to be the best avenue for product improvement. The most significant implication of this aspect to the underground application is the increased weight which is limited by existing ‘Can’ carrying equipment (this assumes that the option of pumping grout into empty cans would violate the patent (Healy, pers commun)).

Since the strength of cementitious products generally increase exponentially with density, a small increase in overall weight may offer substantial increases in overall product strength. In addition to the increase in UCS, the responsiveness of the fill to confinement would also be expected to increase with a denser fill. Consideration of suitable equipment and OH&S issues associated with heavier ‘Cans’ is beyond the scope of this study however communications with colliery personnel suggest that increased weight of up to 100 per cent is feasible with existing handling equipment.

**Field load determination**

Since the stiffness of the ‘Can’ is known and since the strain to yield is known (7500 με), estimation of the load could be obtained through measurement of the total top to base convergence. Given the value of 22.5 mm for a 3.0 m long can, this magnitude of displacement is measurable with a standard (and cheap) convergence pole. Note that the total roof to floor convergence would not provide a suitable ‘strain’ measurement because this value would include the deformation of the packing and closure of gaps between the support and the roof. The reference anchors would need to be attached to the ‘Can’ itself. The strain could also be measured with a simple measurement of the distance between two pins attached to the steel casing.

Once local buckling has occurred, the ‘Can’ exhibits strain hardening behaviour meaning that the load continues to increase with progressive roof to floor convergence. The measurement of load once local buckling has occurred could only be achieved through use of a load cell since the load/strain relationship becomes non-linear.

**Relevance of mini-can tests**

The mini-can tests were able to provide a greater insight into the general mechanisms of load development in the ‘Can’, the findings of which have application to support design using existing products and application to product development. Most importantly, the measurement of the confining stress provided by the steel to the fill was found to be very low (<0.05 MPa for 900 mm diameter cans). The role of the steel cylinder is to carry load rather than provide confinement in the current configuration.

The strength of full-scale products were predicted from the results of the mini-can tests and found to agree well with full-scale testing already conducted. The ability to predict the behaviour and strength of larger scale versions from scaled-down tests indicates that small scale testing would be a suitable tool for product development. The ability to conduct cheaper and easier small scale tests would be expected to accelerate product development and ultimately provide the mining industry with improved support design and product choice.

The most important application of the scaled-down tests is the ability to conduct product development in the laboratory as a first pass, rather than the more hazardous alternative of trial and error in an underground situation.

**Improvements for future scaled-down testing**

The preparation of the mini-cans was conducted according to tolerances applicable to the mining application. Unfortunately the presence of imperfections in the steel cylinders and lack of pre-testing conditioning resulted in an undesirable level of scatter in the strain gauge readings. The size and weight of the mini-cans (>60 kg) made proper sample preparation difficult. The results of the testing program indicate that with appropriate sample preparation, smaller scale cylinders would be suitable. Smaller samples would also make available a greater range of testing equipment (50 t capacity).

Investigations into improved fill properties could occur independently of the steel cylinder. Similarly, improved steel (or non-steel) cylinder strength could occur independently of the fill.

**CONCLUSIONS AND RECOMMENDATIONS**

Testing of scaled-down versions of the ‘Can’ provided a suitable method to predict the stiffness and strength characteristics of full-scale versions in the field. The scaling parameters were a function of:

- strength of the bare (unfilled) steel tube at the onset of local buckling,
- strength of the unconfined fill, and
- contribution of confinement provided by the steel to the fill.

The various load contributions of the steel cylinder, unconfined fill and confined fill were determined for a range of ‘Can’ sizes. The unconfined fill and steel cylinder account for 63 per cent and 35 per cent of the total load respectively for a 915 mm diameter ‘Can’.

The confining stress provided by the steel cylinder to the fill was found to be surprisingly low. The inferred confining stress developed in a 915 mm diameter ‘Can’ was approximately 0.05 MPa, which would account for only two per cent of the total load contribution.

Limited parametric analyses were provided to determine the relative impact of changes to the thickness of the steel cylinder and/or the fill strength. The properties of the fill, in particular the increase in strength with confinement were considered to be areas where significant improvements to overall ‘Can’ strength and stiffness could be achieved. These improvements would need to be assessed against the increased weight and associated handling issues.

Estimation of load in the ‘Can’ in the field could be obtained with reasonable accuracy by measurement of the axial strain at any point up to the onset of local buckling. The measured distance between two pins with a tape measure for example would be suitable.

The scaled-down tests would be a suitable ‘first pass’ to assess the field behaviour of non-steel ‘Cans’.

The equations developed by the civil industry to estimate the buckle strength of thin walled steel tubes were developed empirically using D/t ratios significantly smaller than that...
ACKNOWLEDGEMENTS

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- North Goonyella Coal Pty Ltd,
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- Strata Products,
- Ground Consolidation,
- Coalroc Contractors Pty Ltd, and
- SCT Operations Pty Ltd.

The funding provided by these companies is gratefully received. Pioneer Burrell is kindly thanked for the manufacture of the ‘mini-cans’.

In future tests of this nature, particularly at further reduced tube diameters, stricter control over the residual stresses induced by the manufacturing process should be used. No attempt was made in this program to measure the effects of the imperfections.

In future tests the mini-cans should not be capped since the capping affects the lateral restraint and therefore has an impact on the measurement of confining stresses provided to the fill.

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APPENDIX

INDIVIDUAL STRAIN READINGS
Skew Roof Deformation Mechanism in Longwall Gateroads — Concepts and Consequences

G Tarrant

ABSTRACT

A research project was commissioned by the Australian Coal Association Research Program (ACARP) to improve the understanding of tailgate strata mechanics and to provide a more rigorous engineering basis for tailgate support design. A deformation mechanism termed ‘skew roof’ was defined which relates the regional influence of differential horizontal strata movement (shear) about longwall extraction to gateroads. Confirmation of the mechanism was achieved by field investigations which included measurement of the shear displacement along weak interfaces. Under geological and mining conditions where the skew roof mechanism operated, strata units were found to move progressively further towards the goaf with height into the roof. 3D numerical modelling was used to assess the major geotechnical factors controlling the mechanism and to determine appropriate support strategies within a ‘skew roof’ environment including the role of cables versus standing supports. The skew mechanism is considered relevant to; all roadways within the vicinity of longwall extraction including the faceline itself, chain pillar design, and support design.

INTRODUCTION

A research project (ACARP, in press) was commissioned to improve the understanding of tailgate strata mechanics and to provide a more rigorous engineering basis for tailgate support design. A deformation mechanism termed ‘skew roof deformation mechanism’ was identified which relates the regional differential horizontal movements that occur about longwall extraction to the shear behaviour about gateroads, leading to a range of adverse roadway behaviour. Skew roof has implications for chain pillar design (tailgate positioning) and indeed, all roadways within the vicinity of longwall extraction, including the faceline itself. The implications of the ‘skew roof’ mechanism to tailgate support design are discussed with reference to the relative roles of long tendons versus standing supports and the importance of support positioning.

The work program comprised a combination of observation, field measurement, laboratory investigations and 3D numerical modelling, predominantly at the sponsor mines of the C12006 Project. Field studies were undertaken at the following mine sites in association with ongoing geotechnical investigations:

- Metropolitan Colliery,.pix
- North Goonyella Mine, and
- Moranbah North Mine.

Measurement of the differential horizontal displacement of roof strata about longwall extraction were undertaken at these mines together with measurement of loads developed in standing supports. Field investigations were supplemented with 3D numerical modelling studies where the sensitivity of the factors driving the skew roof mechanism was examined. A detailed description of the C12006 Project results for each mine is provided in the final project report (ACARP, in press). The purpose of this paper is to convey the key findings that are considered transportable to the broader coal mining industry.

Problem definition

The C12006 Project was commissioned in response to an industry demand for more rigorous methods of gateroad support design. This reflected an ongoing occurrence of problematic tailgate behaviour against a background of trial and error approaches to support design. This issue is also echoed in overseas coal mines with Barczak (2003) lamenting that: whilst pillar design practices had improved through use of ALPS, problematic tailgate behaviour was still a major concern in many US longwall mines and that optimization of support design would not be achieved through current trial and error practices.

At two of the sponsor mines in the C12006 Project, the author’s own observations and anecdotal information strongly suggested that horizontal movement of roof strata towards the approaching goaf played a more important role than previously considered. As shown in Figure 1a the specific observations of tailgate behaviour suggested that the immediate roof appeared to have been driven towards the block side. The movement was so severe that the immediate roof material was essentially pulverised and flowed out of the roof space between the installed standing supports as shown in Figure 1b. Anecdotal advice suggested that this style of roof behaviour was evident in various forms in many of the Australian coal mines that experienced poor roof behaviour either adjacent to longwall extraction (travel roads) or during approach of the next longwall (tailgates).

SKEW ROOF DEFORMATION MECHANISM

Proposed hypothesis

The ‘skew roof deformation mechanism’ proposes that under certain circumstances roadways about longwall mining are required, if remaining elastic, to skew. If not for strata softening, rectangular shapes would deform into parallelograms. The propensity to skew is a consequence of a regional gradient of horizontal strata movement towards the goaf, progressively increasing from seam to surface as shown in Figure 2. The affect is regional in that horizontal movements on the surface can extend in the order of kilometres from longwall mining and at seam level the influence can extend over many tens of metres and potentially hundreds of metres.

The direction of the ‘skew’ is a nett influence of the direction from the roadway to the goaf and the direction of the maximum horizontal stress direction. This may actually cause the nett direction of roof skew to be away from the longwall block side but typically the roof is skewed towards the longwall block being extracted.

The initial roof damage associated with the skew mechanism is slip along interfaces between strata units or along bedding within strata units. The slip is not confined to the immediate roof and floor strata but may extend well beyond the riblines. Most importantly, the regional gradients of horizontal movement may continue to occur and cause more deformation of the already softened strata about the roadway. This introduces a component of displacement control on the subsequent deformation of the softened strata which may impact on the support strategy. The key factors driving the skew roof mechanism are considered to be:

1. MAusIMM, Senior Strata Control Engineer, SCT Operations Pty Ltd, PO Box 824, Wollongong NSW 2520. Email: gtarrant@sctaust.com
the magnitude of the vertical and horizontal stresses;
the shear modulus of the strata pile (shear deformability); and
the extent of overburden bridging.

Major factors that are considered to influence the extent of roadway damage include:

- the proximity of the roadway to longwall extraction;
- the presence of weak interfaces in the vicinity (several metres) of the roof and/or floor;
- installed artificial support; and
- strata damage about the roadway experienced on initial driveage.

Clearly there are many other factors that may also influence the extent of roadway damage including those factors that impact on roadway damage on initial driveage and all operational factors that impact on the load distribution about the longwall such as powered support capacity and yield setting, etc.

**Relationship of skew roof to vertical and horizontal stress changes**

The skew roof deformation mechanism overprints the roof deformation mechanisms that are attributed to vertical and horizontal stress changes about longwall extraction. The potentially adverse impact of high vertical pillar loading on roadways has been well established through empirical studies such as ALPS (Mark, 1990, 1992, 1999) and ALTS (Colwell et al., 1999). Mills and Doyle (2000) discuss the adverse consequences of high vertical loading on roadway behaviour at Dartbrook Mine using rock mechanics principle centred on the Poisson Effect. Essentially the vertical compression of the pillar results in an increase in horizontal stress in the roof and floor strata. In an elastic environment this is of the order of 33 per cent of the vertical stress increase for most non-coal strata and up to 50 per cent for coal strata (Mills and Doyle, 2000). The increase in horizontal stress may result in overstressing of the roof and floor strata.

The impact of horizontal stress on roadway damage has been well documented (Siddall and Gale, 1992). Essentially elevated in situ horizontal stresses may result in overstressing of the roof and/or floor material and contribute to shear along bedding as stresses rotate about the roadway opening. Mine layouts are generally designed to minimise the concentration of horizontal stress about longwall extraction with typically the most favourable extraction orientation subparallel to the maximum horizontal stress direction.
The skew roof mechanism overprints the affects of horizontal stress damage. As stated previously, strata that is already damaged about the roadway is still subjected to the regional differential horizontal strata movements associated with longwall extraction. Under these circumstances the softened strata about the roadway could be considered ‘slaved’ to the deformation of the host strata. The magnitude and direction of the pre-mining horizontal stress also has a major impact on the direction of ‘skew’ and the extent to which the skew process impacts on the roadway as will be discussed in greater detail.

In summary the proposed skew roof deformation mechanism operates in conjunction with those deformation mechanisms attributable to changes in the vertical and horizontal stress components. The skew process relates to the rotation of the principal stresses out of the horizontal plane.

Supporting data for differential horizontal movement

Surface subsidence monitoring
Reid (1998) measured horizontal movements at the surface of up to 25 mm approximately 1.5 km from longwall mining in terrain surrounding the Cataract Dam where mining had occurred at depths up to 500 m. Reid (1998) also noted that:

horizontal movements are typically at least as great as the vertical component, that the maximum horizontal movement occurs soon after undermining and that the movements are generally directed towards the goaf.

Holla (1997) measured vertical and horizontal surface movements associated with longwall mining in flat and high relief terrain in the Newcastle Coalfields. Horizontal movements of over 260 mm were recorded at distances of half the mining depth and whilst not specifically discussed, the figures presented indicated horizontal movement of at least 20 mm up to 1 km (the limit of the subsidence line) from longwall mining.

Hebblewhite et al (1999) noted that monitoring of horizontal surface movements associated with longwall mining at Tower Colliery (450 m deep) recorded horizontal displacement of 60 mm at 1.5 km from longwall extraction.

The reader is directed to these texts for more detailed explanation of the impact of surface horizontal movements however the key point is that significant horizontal movements have been recorded at the surface at great distances from longwall mining.

Numerical modelling
3D numerical modelling has been conducted for three of the sponsor mines associated with the ACARP C12006 Project as listed previously. The modelling approach and detailed discussion of results is provided in the final report (ACARP, in press); however, Figure 3 illustrates a typical cross-section (example from Moranbah North Mine) showing contours of horizontal displacement towards the adjacent goaf; 225 m depth. Note: gateroads shown for reference only.

**FIG 3 - Contours of horizontal displacement towards the adjacent goaf; 225 m depth. Note: gateroads shown for reference only.**

![Displacement Contours](image)

**FIG 4 - Relative movement – Travel Road, Longwall 9, Metropolitan Colliery.**

- a) Longwall approaches immediate roof
d) roof’s movement towards roadway opening (Poisson effect)
- b) Adjacent to Longwall goaf
  sandstone moved further than m dstone m dstone moved further than coal

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Field measurements at seam level

The relative horizontal movement of strata about gateroads adjacent to longwall extraction was conducted at the three sponsor mines adjacent to longwall extraction (full side abutment loading) for each of the mines and also under tailgate loading at Metropolitan Colliery.

At each monitoring site the array of field instrumentation included shear strips installed at 45° over the riblines. Each shear strip comprised 72 strain gauges (36 each side) at 50 mm intervals on a stainless steel bar over a total length of 2.0 m. The bar was sealed within a rectangular housing and grouted into a 60 mm diameter hole. Shear displacement of the strata causes the bar to bend and the magnitude of shear displacement is calculated through the differences in strain developed either side of the bar. Prior to installation of the shear strips, candidate locations for shear displacement were identified through roof coring and in each case a clear candidate was identified.

The location of the shear strips in relation to the roof geology at Metropolitan and the sense of movement during approach of the adjacent longwall and then behind the goaf are shown in Figure 4. For the sake of brevity, only the shear strip data for Metropolitan Colliery is provided in the Appendix (and Figure 5). The complete shear strip data in terms of the strain changes measured and the cumulative displacements for each of the mines is provided in the final project report (ACARP, in press).

The shear strip data clearly shows the presence of a shear plane (indicated by the ‘Z’ shape) that developed with extraction of the adjacent longwall. The shear horizon coincided with the candidate location identified from roof coring.

During approach of the adjacent longwall the sense of shear was lower roof towards the roadway centreline which is consistent with that expected from vertical loading and flexure of the lower roof layer as shown in Figure 4. Note that the sense of shear between the lower and upper roof layers is opposite on each side of the roadway at this stage. As mining drew level and passed the monitored site, the sense of shear reversed on one side of the roadway such that the sense of movement was consistent with the upper layer moving further towards the adjacent goaf compared with the immediate roof layer. This was consistent with the skew roof mechanism. At this stage in the mining cycle the magnitude of shear displacement was approximately 5 mm and no discernible roof damage was observed.

The site continued to be monitored during approach of the next wall and the sense of shear continued (upper roof towards the adjacent goaf) until a clear reversal became evident on the block side when the next longwall approached between 52 m and 36 m from the monitored site. The reversal was clearly detected as shown in Figure 5 where Figure 5a illustrates the strain changes from installation and Figure 5b illustrates the strain changes and sense of shear using the readings when the approaching wall was >50 m from the site as the reference.

![Figure 5](image-url)
The shear strip results from the central heading adjacent to longwall extraction at Moranbah North Mine are summarised in Figure 6 which also showed a similar style of shear behaviour as that evident at Metropolitan Mine. In this case shear movement occurred along the Rider Seam/stone interface. Initial shear displacement on approach of the longwall and at least 81 m behind the wall was dominated by vertical loading, roof flexure and associated Poisson effects and then between 81 and 195 m behind the adjacent goaf, the strata above the shear plane moved towards the goaf relative to the strata below on both sides of the roadway. The shear movement was greater on the block side which is also consistent with the inferred rotation of the principal stresses as shown in Figure 2.

The shear strip results from North Goonyella associated with adjacent longwall extraction are summarised in Figure 7. The data suggested a reversal in the sense of shear across the thrust plane consistent with skew roof behaviour after the wall had passed by 75 m however the results are considered to be influenced by general softening about the roadway and are far from convincing. It is considered possible on the basis of field observations and numerical modelling that the shear inferred about the roadway may reflect the development of high angle zones of bedding developed at or beyond the riblines. The remaining shear strips installed across the coal/stone interface were consistent with behaviour expected from the Poisson effect.

**Field results summary**

Each site clearly detected the presence of shear along weak interfaces about the roadways associated with adjacent longwall extraction. This in itself has implications for both roadway and pillar behaviour however the key objective of the fieldwork was to establish whether or not strata units higher into the roof moved further towards the goaf in response to a regional gradient of horizontal movement from seam to surface (skew roof mechanism). The skew roof mechanism was convincingly indicated by the shear strips at Metropolitan Colliery on approach of adjacent longwall extraction and then under tailgate loading conditions. At Moranbah North Mine the skew behaviour was detected but much later relative to adjacent longwall extraction (>81 m behind the goaf). At North Goonyella Mine the results were not conclusive in relation to the operation of the skew roof mechanism and it is considered more likely that the shear detected was a consequence of general softening and mobilisation of structured ground.
DISCUSSION

Factors influencing skew roof mechanism

The proposal that progressively increasing horizontal movement towards the goaf from seam surface may impact on gateroads was confirmed through field measurement. The regional horizontal movements imposed relative movement about the roadways such that the roadway roof moved towards the goaf more than the floor and layers higher into the roof moved further than immediate roof layers, all other things being equal. The extent of the relative movement and consequential slip along bedding varied considerably between the three mines and the following discussion presents the findings of work conducted to better understand the underlying factors driving the skew roof process.

The driving force for shear displacement is shear stress. The shear stress generated in the plane of bedding is the driving force behind slip along bedding.

The three mines represent a range of mining and stress environments as summarised in Table 1. The skew roof behaviour was greatest at Metropolitan which implied that either depth and/or horizontal stress may have been contributing factors. Moranbah North and North Goonyella represent similar mining geometries with the major differences being that the horizontal stress is somewhat higher at Moranbah North and also the orientation of the maximum horizontal stress with respect to mining is significantly different. The range of variables in the field sites also included other factors such as average rock stiffness of the strata pile from seam to surface, rock stiffness contrasts of the immediate roof strata, presence of structure and variation in the strength of interfaces.

3D numerical modelling of the three sponsor mines provided a measure of the shear stress that would have been imposed on the gateroads during extraction of the adjacent wall (under full side abutment loading). The shear stress obtained from modelling represents the nett outcome of the competing influences listed above. It was considered important to use the 3D code so that the effects of longwall extraction oblique to the principal stresses would be captured. Skew of the rectangular roadway can be visualised as a combination of skew both across and along the roadway and each of these components contributes (by vector addition) to shear stress in a given plane.

Figure 8 is a contour of total shear stress resolved in the plane of bedding (horizontal in these cases) for the same modelled scenario as that shown in Figure 3. The 1 MPa contour is highlighted and the area containing shear stress along bedding greater than 1 MPa is shaded. Any roadway positioned within the shaded region would be expected to experience shear along bedding to some extent (clearly this depends on the shear strength of the interface). The actual position of the gateroad in this case is also shown in the figure and clearly the likelihood of slip along bedding was high.

The distribution of shear stress along bedding away from the goaf is summarised in Figure 9 which also includes the distribution obtained from modelling conducted for Metropolitan.

![Figure 8](image1.png)

**FIG 8 - Contour of shear stress in the plane of bedding under full side abutment loading – Moranbah North, 225 m depth. Note: roadways shown for reference only. Shaded area represents region in which high potential exists for shear along bedding in the roof and floor of a roadway.**

![Figure 9](image2.png)

**FIG 9 - Modelled shear stress on bedding versus distance from goaf edge – side abutment loading.**

### TABLE 1

<table>
<thead>
<tr>
<th>Mine</th>
<th>Depth of cover (m)</th>
<th>Longwall width (m)</th>
<th>Pillar width (centres) (m)</th>
<th>Tectonic setting</th>
</tr>
</thead>
<tbody>
<tr>
<td>Metropolitan</td>
<td>500</td>
<td>155</td>
<td>40</td>
<td>Relatively high horizontal stress environment. 30° maingate stress concentration.</td>
</tr>
<tr>
<td>Moranbah North (three heading layout)</td>
<td>225</td>
<td>250</td>
<td>30 and 25</td>
<td>Relatively moderate horizontal stressfield. 0 - 25° tailgate stress concentration.</td>
</tr>
<tr>
<td>North Goonyella</td>
<td>250</td>
<td>250</td>
<td>35</td>
<td>Low to moderate horizontal stresses. 30° maingate stress concentration.</td>
</tr>
</tbody>
</table>
Colliery and North Goonyella Mine and the respective gateroad positions. The figure shows that the magnitude of shear stress imposed on Metropolitan gateroads adjacent to longwall extraction was significantly higher than Moranbah and North Goonyella. Whilst the Moranbah North and North Goonyella distributions were similar, the central heading of the former mine was located within a region of higher shear stress. The modelled distributions suggest that slip along bedding would be expected about any roadway located within 40 to 50 m of the goaf edge at North Goonyella and Moranbah North and over 80 m from the goaf edge at Metropolitan Colliery!

To better understand the factors resulting in the difference between shear stresses at different mines, a desk top study was undertaken to examine the impact of depth, rock stiffness (shear deformability of the strata) and in situ horizontal stress on shear stress about longwall extraction. The study was conducted using the 2D Flac code and therefore no ‘out of plane’ shear stresses were modelled. The strata section used was a generic case, simplified such that the only two rock types were coal and another rock type whose properties were varied. The pre-mining horizontal stress was input according to Equation 1, which scales the horizontal stress according to depth and a tectonic stress component. The study is described in detail in the ACARP C12006 Final report (ACARP, in press) and only the key findings are discussed here.

\[ \sigma_{\text{Hotal}} = \sigma_v \times \left( \frac{v}{1-v} \right) + TSF \times E \]  

where:
- \( \sigma_{\text{Hotal}} \) is the horizontal stress
- \( \sigma_v \) is the vertical stress
- \( v \) is the Poisson’s Ratio
- \( TSF \) is the tectonic stress factor
- \( E \) is the Young’s Modulus

Figure 10 shows the shear stress in the roof of a hypothetical roadway located 40 m from the goaf edge for the generic case modelled for a range of depth and rock stiffness and tectonic stress factors (TSF). The first point in each curve is a TSF of 0.2, which would represent a very low tectonic stress environment and the last point is a TSF of 1.4, which represents a high horizontal stress environment. The figure shows that:
- for a given rock stiffness and depth, shear stress generally increases with the magnitude of the pre-mining horizontal stress;
- for a given depth and horizontal stress, shear stress increased with shear modulus with much greater sensitivity in the E = 5 to 12 range compared with E = 12 to 20 range; and
- for a given rock stiffness and pre-mining horizontal stress, shear stress was not related to depth of cover.

The sensitivity study highlighted that the interrelationships between shear stress and depth of cover, rock stiffness and horizontal stress are complex and that evaluation of the propensity of any given environment to exhibit skew roof behaviour requires a numerical approach to gain an understanding of the interaction between competing influences.

One of the important findings of the sensitivity study was the lack of a clear relationship between potential skew roof deformation and depth of cover. This suggests that problematic gateroad behaviour can occur at depths below that suggested from analysis of vertical loading alone and conversely that increased depth may not necessarily make skew roof behaviour worse.

**Skew roof at the tailgate/faceline corner**

The preceding discussion developed the general concepts of the skew roof behaviour for the relatively simple extraction geometry of an adjacent goaf ignoring end effects.

The extraction geometry at the tailgate corner is considerably more complex however the general concepts already developed still apply. The 3D numerical modelling work conducted for the three sponsor mines indicated that the direction of the shear stresses resolved in the plane of bedding (the skew direction) was influenced by both the direction to the approaching goaf and the pre-mining horizontal stress direction. This is conceptually illustrated in Figure 11 for a range of extraction orientations with respect to the maximum horizontal stress. It is interesting to note that whilst there would usually be a component of skew movement towards the block side, under some circumstances there can be a component of movement away from the block side.

Figure 12 is a summary plot showing the distribution of shear stress along bedding in the tiltage roof versus distance to the faceline for the three sponsor mines. Based on the general assumption that 1 MPa shear stress would be sufficient to generate shear along bedding in the roof or floor of a roadway, the distributions suggest that shear behaviour would be expected at Metropolitan and Moranbah North with the former being more extensive. In contrast, the shear stress about the tiltage at North Goonyella would not be expected to be as high. This is mainly a consequence of the magnitude and orientation of the maximum horizontal stress direction.

**Style of damage from skew roof behaviour**

The preceding discussion presented some major factors contributing to the generation of shear stress about roadways in the vicinity of longwall extraction. The likelihood of the imposed shear stress causing slip along bedding and consequential adverse roof behaviour is itself dependent on many factors. Clearly the strength of the interfaces is a major factor. The following discussion is based on the assumption that a weak interface is present within the immediate 5 m of roof or floor and that the shear stresses are such that the skew roof mechanism is operating.
a) Maingate stress notch - high component of movement towards the block side.

b) Maximum $\sigma_H$ parallel to heading - high component of skew along the gateroad
   (very bad for cut-throughs and faceline
   - small component across the gateroad

c) Tailgate notch - potential component of movement away from block side

Fig 11 - Conceptual model of skew direction caused by direction of extraction with respect to the maximum horizontal stress orientations.
The key aspects of damage associated with the skew roof mechanism:

- increased shearing along weak interfaces in the roof beyond the riblines on the goaf side of the roadway;
- increased shearing along weak interfaces in the floor beyond the riblines on the other side of the roadway;
- increased roof damage typically on the goaf side of the roadway (naturally pre-existing roof damage may continue to focus subsequent damage); and
- increased floor damage on the same side of the roadway as the roof damage.

Potentially the most important characteristic of the skew roof mechanism is the perfectly plastic behaviour (displacement driven) of the softened strata about the roadway. The strata surrounding the roadway will differentially move towards the goaf whether or not the roadway is present. If the movement was simple translation without a shear component, then an observer whether or not the roadway is present. If the movement was simple translation without a shear component, then an observer would scarcely notice however the shear deformation. This is achieved by early installation, tight packing and pre-stressing with inflatable packers.

**Cable bolts should be used to assist with maintaining roof integrity on the block side however their ability to limit lateral strata movement associated with skew roof is considered to be limited.**

**FIG 12 - Distribution of shear stress about the tailgate versus distance from the longwall faceline.**

**IMPLICATIONS FOR SUPPORT DESIGN**

Tailgate roadway behaviour is sensitive to a combination of vertical, horizontal and under some circumstances shear stress changes (skew roof) associated with longwall extraction. The roof strata moves laterally towards the adjacent goaf under side abutment loading and under tailgate loading the movement changes direction towards the approaching goaf. In addition to this movement, the vertical loading of the ribs causes rib softening which increases the effective span of the roof and floor and also induces further lateral movement in the roof and floor according to the Poisson effect.

One of the key objectives of the support design should be to reduce the lateral strata movement towards the block side as far as possible. The ‘collision’ of the immediate roof strata against the block side causes roof and floor damage by itself and exacerbates any other primary drivers of roof damage such as horizontal stress increases or elevated pillar loading. If the lateral strata movement cannot be reduced sufficiently, then protection of the block side should be considered, potentially accepting increased damage on the chain pillar side of the roadway. This strategy has proven to be an effective method to manage skew roof style of behaviour at Metropolitan Mine. The method seeks to prevent damage propagating from the tailgate along the faceline and provides a stable section of roadway for a second means of egress.

**Protecting the block side**

The positioning of standing support in a line rather than in a staggered pattern is considered to be an effective means of predisposing the roof or floor damage to occur on the chain pillar side rather than the block side. Essentially the lateral movement of the immediate roof strata ‘collides’ against the artificial barrier presented by the line of standing supports rather than against the block side. The following aspects are critical to achieving an effective artificial barrier:

- The standing supports must be placed close enough to interact as a pattern. This typically ranges from 3 to 5 m but should be confirmed for the conditions specific to each mine.
- The integrity of the immediate roof must be maintained as far as possible. This impacts on the primary roof bolt density. It is considered unlikely that a four-bolt pattern would maintain an acceptable level of roof integrity under skew roof conditions.
- The integrity of the roof skin is critical. Strong mesh is considered an essential component for this strategy with the use of suitable bolt plates such that high collar loading does not result in premature failure of the bolt/plate/mesh system.
- The line of standing supports should be biased towards the block side.
- The standing supports should be engaged as early as possible by the roof to floor convergence. This is achieved by early installation, tight packing and pre-stressing with inflatable packers.
- Cable bolts should be used to assist with maintaining roof integrity on the block side however their ability to limit lateral strata movement associated with skew roof is considered to be limited.

**Role of cable bolts**

Cable bolts are considered to be an excellent product to reduce strata dilation, particularly when this occurs above the bolted horizon. The ability of cable bolts to reduce lateral shear displacement is considered to be limited. Unfortunately under circumstances where skew roof deformation is active, a simple substitution of cable capacity for standing support capacity would be an inappropriate design choice.

It is highly recommended that standing supports are used to control the lateral strata behaviour or to reduce its impact rather than cables. Cable bolts would perform an important function on the block side by maintaining roof integrity. When used in conjunction with standing supports, the cables on the block side are theoretically operating in a zone of reduced shear displacement which would enhance their longevity and performance.
a) Can A approximately 0 m o tbye

b) Can approximately m o tbye

c) Can C approximately m o tbye

d) Can D approximately m o tbye

FIG 13 - Views of tailgate looking inbye – roof conditions on block side of supports significantly better than chain pillar side.
Future support improvements
Numerical modelling conducted as part of this project sought to determine the influence of support aspects such as positioning, stiffness, strength and timing of installation. This work will be reported in more detail elsewhere however the results suggest that a substantial increase in support stiffness and capacity (to at least 350 tonnes and potentially up to 500 tonnes) would be required to have an impact on the skew roof mechanism. The impact of such support capacity would need to be evaluated against other aspects such as supports punching into the roof.

Significant improvement to the stiffness of existing support systems would be expected to result from active setting of the supports against the roof. Use of inflatable packers would be expected to offer significant benefits in terms of support performance compared with existing methods of base construction. Pre-stressing the supports in this way also removes that component of roof to floor convergence required to ‘seat-in’ the support. This was found to be in the range 20 to 50 mm which would allow significant loss of roof strength prior to the supports being effectively engaged.

CONCLUSIONS/RECOMMENDATIONS
A roof deformation mechanism termed ‘skew roof’ has been investigated through observation, field measurement and 3D numerical modelling. As the term suggests, the immediate roof layers in the vicinity of longwall extraction may move further towards the adjacent or approaching goaf relative to the floor. Similarly, higher strata layers move further than layers closer to the roof. In extreme cases the immediate roof layer is ‘driven’ towards the goaf under displacement (unstopable) control.

The skew roof mechanism is driven by the rotation of the principal stresses out of the horizontal plane about longwall extraction, thereby imposing excessive shear stress on (near) horizontal bedding. The key factors driving the skew roof mechanism are considered to be:

- the absolute and relative magnitudes of the vertical and horizontal stresses;
- the shear modulus of the strata pile (shear deformability); and
- the extent of overburden bridging.

The skew mechanism overprints other influences such as vertical loading of chain pillars and associated Poisson effects and exacerbates the affects of horizontal stress changes.

Control of tailgates subjected to skew roof behaviour is currently limited to protection of the block side, considered to be the most critical area to safe egress of personnel and manageable faceline conditions. Standing supports are considered to be the ‘front line’ support strategy with emphasis on appropriate positioning (biased to the block side), interaction as a pattern and maximising effectiveness through early placement and pre-stressing. Primary bolts, roof mesh and long tendons have important roles to play in maintaining the integrity of the immediate roof layers to allow the standing supports to interact effectively.

However, significant increases in the existing strength and stiffness of standing supports is considered necessary to achieve better control of problematic tailgates.

The determination of the potential impact of skew roof behaviour and development of an appropriate support strategy remains a ‘horses for courses’ proposition, requiring observation of strata behaviour, field measurement of support loading and strata interaction and numerical modelling which should be 3D to properly simulate the full 3D stress changes.

The skew roof mechanism should also be a consideration in pillar design, mine layout and the potential impact on the longwall faceline itself. Computational advances allow the reasonable 3D simulation of the tailgate environment and evaluation of any conceivable support strategy. This provides a rigorous basis to optimise pillar width against support effort.

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REFERENCES


APPENDIX
SHEAR STRIP RESULTS – METROPOLITAN MINE
Note that the reference location (zero displacement) is the base of the shear strip however since the absolute position of the shear strip is not known, the ‘Y’-axis in the cumulative displacement plots could be moved to the right or left. This does not alter the sense of shear that would affect the absolute position of the shear strip itself. In other words, the absolute location of the shear strips is not known relative to a fixed reference.
**FIG A1 - Cumulative displacement – chain pillar side during extraction of Longwalls 9 and 10.**

**FIG A2 - Cumulative displacement along shear strip – block side during extraction of Longwalls 9 and 10.**
Fig A3 - Strain change on chain pillar side after extraction of Longwalls 9 and 10.
Management of Change — The Transformation of UK Coal Project 105

K J Irving

ABSTRACT
In the UK during the 1990s the energy companies had developed a policy of ‘Dash for Gas’, building large natural gas power stations to produce more than 30 per cent of the UK’s electricity demand. The price of imported thermal coal was dropping and the electricity generators were putting pressure on the UK coal producers to produce coal at a reduced cost. Imported coal from Columbia, South Africa, America and Australasia had increased their market share in thermal coal sales to over 50 per cent of a 55 Mtpa market. The pressure on the UK coal industry to reduce costs and improve productivity levels from an ever decreasing resource was high. Companies were benchmarking their operations on the imported coal producers and had to realign their organisations, productivity levels and costs to world best practice, not just technically, but in world class organisational management techniques.

INTRODUCTION
In the 2001 UK Coal, at the time Europe’s largest independent coal operator, announced its intentions to embark on a management of change process to transform one set of beliefs and values to another. The case study will describe one of the most exciting and stimulating periods that UK Coal PLC experienced since the industry was privatised in 1994. The focus of this paper is on the transformation of the deep mines division of the company, but the whole program brought about change throughout all the functions within the group, from marketing through purchasing to the coal face operations.

BACKGROUND
UK Coal was born out of the privatisation of the coal industry by the Conservative government in late 1994. Richard Budge, an entrepreneur with a background in open cut coal contracting, leased in 1994 the year before privatisation, the licence to mine coal at three underground closed mines. In 1994, on the back of successfully re-opening these mines, he took his company to a public offering, and financed the buying of the English Coalfield at a cost of £ 814 million. In 2001 the company changed its name from RJB Mining to UK Coal PLC.

In 2001 UK Coal was Europe’s largest independent coal mining company, and 14th largest world producer. In 2000 the company produced 19.1 Mt of saleable coal, mainly thermal, 15.2 Mt from deep mines and 3.9 Mt from surface operations. Employing some 7200 people with an annual turnover of A$ 1700 million, production coming from 13 deep mines and 13 surface mines.

In the early years the company profited from prices agreed under the nationalised British Coal Company and the realisation of the stocks bought under the privatisation agreement.

In 1997/1998 the contract prices were floated, to fall in line with the competition and the prices reflected the direct competition from imports.

In 1995 the company held 50 per cent of the market in thermal coal in the UK; the market share has since fallen to less than 20 per cent. Productivity had barely improved. The cost of operations had increased and the unit cost of production had steadily risen.

In 2001 the main board of the PLC decided to launch an initiative named Project 105 (targeting the cost of production to £ 1.05/gj) and direct a complete transformation of the company and how it operated.

This program of change entailed a detailed analysis of all facets within the company from mining operational efficiency, purchasing consumables to marketing.

The objectives of the program were to:

• transform UK Coal into a world class coal mining company;
• produce coal at less than £ 1.05/gj by the end of 2003; and
• build a long-term future for the UK coal industry.

The program had five key elements in the deep mines section:

• drive costs down through productivity improvements and cost;
• align the organisation structure of deep mines, HQ, and surface operations;
• develop a management process that would become the UK Coal way of management and a model for others to follow;
• to achieve mining excellence in production, development, installation and salvage; and
• expand capacity by increasing production time through changing planned shifts with the agreement and support of the workforce.

The program was to take two years to implement and it began with a phased approach (see Figure 1).
After the initial diagnostic stage there were five key projects to which the program aligned itself for detailed design and implementation:

- deep mines turnaround,
- purchasing,
- surface mines improvements,
- market revenue growth, and
- business control.

The change program that this paper will focus on will be the deep mines operations as it accounted for 80 per cent of the coal operations.

DEEP MINES TURNAROUND

The 13 deep mines were mainly longwall retreat with one mine using room and pillar methods. Most mines were operating at depths of around 600 m with some operations nearer 1000 m.

During the initial three months of diagnosis there were five areas, which became apparent needed to be changed:

- deep mines organisation structure,
- HR initiatives,
- productivity improvements,
- maintenance process,
- focussed on improving development performance:
  - by detailed design and risk management, and
  - straight line drivages.

Deep mines organisation structure

Before the program took place the organisation structure consisted of a managing director, and a director of mining to whom 13 mine managers reported directly.

The initial step was to reorganise these 13 mines into three distinctive groups comprising of:

1. long life mines focussing on continued capital investment and improving productivity;
2. mines who were marginally profitable or loss-makers and needed turnaround into profitable entities; and
3. short life mines that would close in the near future, mainly consisting of the Selby Coalfield pits.

The decision behind this grouping was to focus a specific strategy to each of the groups, ie investment, turnaround, and cash cows. Each group was then managed by a small team – consisting a group director, accountant, engineer, and planner. This enabled the mine manager to focus on costs and production and the group to focus on mid- to long-term investment planning and engineering. The group team were heavily involved in the roll out of the new initiatives from the change program.

HR initiatives

A new approach to incentives was introduced for all the deep mines, the aim being to align, simplify and standardise agreements. Before, the mine managers had agreed local incentives and this had become a divisive way for the unions and the workforce to play agreements off against different mines and, of course, choosing which agreement at which mine suited them best for their negotiations.

Flexible working patterns, eg 24/7 (most mines in the UK worked five days with overtime at weekends) were introduced to improve – productivity where there was perceived to be a significant benefit. For example at some mines with shaft capacity constraints like Welbeck Colliery, there would have been no significant benefit increasing the working shifts, whilst at Maltby mine there was a benefit and 24/7 was introduced successfully.

Management process (productivity, maintenance and development)

To target the productivity improvements UK Coal introduced a new management process built around the well known cycle of:

- plan,
- do (action),
- review, and
- intervene.

- Take appropriate direct action to change or accelerate process
- Set appropriate targets
- Regularly monitor variances
  - Output
  - Actions versus plans
- Decide necessary interventions and changes to plan
- Create and communicate effective plans to achieve accountability
- Run mines according to plan
- Adjust plans/actions to respond to unexpected events

Fig 2 - Management process.
The process was about integrating the three main functions of mining, development-production-maintenance. Mining has never been easy and never will be and there have always been major roadblocks to introducing management systems, which were extensively used in manufacturing for example, TQM, TPM. The main difference between manufacturing and mining in terms of input-process-output is the high level of uncertainty and risk, and the significant physical distance between management and the operations. To tackle the uncertainty there needed to be good planning.

UK Coal took a normal management process ‘plan, do, review, intervene’ and adapted it to create a robust, effective way of managing the business on a daily basis to improve profitability.

The planning cycle

The essence of the new process was a robust planning cycle as shown in Figure 3.

Most coal mining companies have a process for steps one to three in Figure 3. The five year plan, the annual budget and detailed project plan. The project initiatives would cover for example face transfers, development heading start ups and major conveyor drive installations. Where the UK Coal process was to differ would be the application of the same detailed planning into the daily operational cycle. This involved the introduction of a planning cycle for the mine management starting with a four weekly rolling plan, a weekly plan down to a daily plan. The ultimate aim to improve productivity was through better communications, good planned maintenance, and resolving any resource or transport constraints between the many teams in the mines.

To ensure good planning one must have high quality relevant data and good interpretation of the information.

The planning process must address the risks and plan for contingencies. Unforeseen events must be considered. The process should allow deviations from the plan to be recognised and dealt with at the lowest possible level.

The Scottish poet Robert Burns wrote in one of his poems:

But, mousie, thou art not alone,
In proving foresight may be in vain,
The best-laid schemes of mice and men
often go astray,
And leave us nothing but grief and pain,
For promised joy

Plans will change. In fact, in the uncertain environment of coal mining – they (the plans) must change. A good, robust management process allows active planning, rather than planning by reaction, to ensure that desired outcomes are achieved.

The second key to good planning is to ensure that those responsible for implementing the plan have ownership. Then they must be held accountable for the outcome. Within the new process, shift bosses and shift supervisors were involved in the process and this had a major impact on results due to increased ownership.

The process ensured that the daily plans and weekly plans were printed and delivered to the key underground operators by the afternoon shift after the plan had been finalised. Each shift had its tasks written out and were held accountable for the delivery of their planned performance.

Efficient, cost effective longwall mining means using the high capital value equipment for as many hours in a day as possible, for every day over the length of the panel. One example of exploiting good planning is integrating the maintenance policy into the production cycle and not adopting the attitude ‘the belts are stood for a few hours, let’s take the opportunity to do some maintenance’.

Who has ownership of maintenance? The longwall superintendent or the engineering department? Some would say the whole team has ownership. Under a good management plan, only one person has ownership, and that is the longwall superintendent. He also carries the burden of accountability.

In a target led organisation when making plans, there needs to be constructive challenge from immediate supervisors or managers. There needs to be a healthy challenge between those who create the plan and those who manage the plan. There must be a focus on being ambitious yet challenging. If too aggressive the personnel carrying out the actions will feel failure; too hands off, and the plans will fail.

A good business plan defines not only what you propose to achieve, but how you are going to achieve it. It must be a living document and the driving force behind the business. The documentation that was developed were four weekly rolling plans, weekly plans and daily plans:-

Four weekly rolling plans

The underground monthly plans looked at the non-routine process activities and exceptional items, such as substation or power moves and emptying conveyor storage loops. Manpower requirements were estimated for jobs that were likely to require...
extra men and by looking at the plans, conflicts of resources were easily identifiable and conflicting issues resolved to ensure efficient use of the workforce.

The phased performance was calculated for each shift, based on planned downtime and assumptions from historical face performance. The target for the team was set and an individual nominated to be responsible to make the work happen on time and to budget, or better still to beat the target.

The surface plans were similar for the coal preparation process, maintenance, mobile plant and any other surface operations.

**Weekly plans**

These meetings were organised to be held every Friday and was used as a tool to integrate the monthly planned work into weekly tasks, but in greater detail. Other tasks, which had been highlighted from a dedicated team dealing with detailed analysis of current and past performance from the longwalls and the developments, were then integrated into the weekly activities. For example, one longwall was suffering from poor ‘t’ junction roof conditions and required extra support. The face teams came up with a practical solution for delivering the extra support. This was then added to the daily transport plans, and also incorporated into the daily face management plan. This improved turnaround times at the face end by over five minutes per turnaround; significant improvement.

**The daily plan**

The next part of the process were the daily plans. These were made at a formal meeting arranged early each morning to review the previous days performance and plan the next daily shifts work load. The tasks were separated by discipline (mechanical, electrical and mining activities). The reasoning behind this was to ensure there were no conflicts between the engineering and mining requirements and resources required.

The daily plans produced at each mine included transport plans, longwall production, and development production. This regime of planning monthly, weekly and daily was in essence the main change to the planning cycle. This level of detail was applied by UK Coal management teams normal applied to projects such as the face transfers began to show improved results; downtime due to poor maintenance was reduced, development productivity improved and the shearsers uptime became noticeably greater leading to improved productivity.

**Action**

Execution of a plan is simple in theory yet hard to achieve in practice. Why? Well this varies from company to company. Execution relies on people; how engaged they are in the organisational process, and their level of involvement and commitment. If the people aren’t engaged, involved and committed, the plan will fail, and you will never fulfil the potential of the business. This applies from the very top, the CEO, all the way down to the supervisors. Engagement and commitment is shown by example. Another adage tells us that ‘the least we are prepared to accept is the most we will ever achieve’. If company leaders do not show exemplary standards of commitment and engagement, why should those below them?

By the same token, the company must ensure that they have the best people. If you don’t get the people process right, you will never fulfil the potential of the business. Action depends on leadership at all levels. This requires a process that ensures that the right people are in place to execute the plan, since without them you may have no hope of executing it.

A robust people process provides a powerful framework for determining the organisation’s talent over time and for planning the actions required to meet those needs. The process must be based on an understanding of the needs of the company, developing leadership at all levels, and succession planning in depth. It is clear that the operations of the human resources department must be integrated into the overall management process.

**Review**

Why review? It could be a pointless exercise building plans if no time were invested in reviewing performance against plan in a logical systematic manner. The same emphasis and time is needed on the reviewing cycle as in the planning. There is a need to assess all aspects of performance; both the hard metrics (tonnes and metres) as well as the compliance to planned jobs. How much time is spent in turning around the shearer at the longwall face end? How good is the roof support system in the gates to overcome front abutment pressures to ensure continuous mining?

Longwall operations these days produce a forest of information. In UK Coal another change was to introduce dedicated personnel who collated the data, analysed the data and then problem solved with the management team to look for improved performance. The team usually consisted of two personnel; one who was focused on longwalls and the other person dedicated to development inventory improvement.

The new reviewing process revolved around accountability from the manager to the longwall supervisors (deputies). UK Coal set a formal system for review, to check the plans and find the right challenge through:

- manager’s meeting,
- variance meetings, and
- delay analysis meeting.

The criteria being to check the plans and find the right challenge to the process.

Coal mining, surface or underground, has four main metrics; safety, tonnes, development rate and costs. These are hard metrics – they can be measured accurately, and these metrics and their subsidiaries must be the key focus of a target led organisation.

Good data is genuine data. It is real, validated and consistent in quality. When data is genuine, management can be confident that decisions are being made on the basis of relevant information.

Most modern longwall mines generate abundant data, usually through the output of the various automated monitoring systems and software packages. This information often encompasses mine planning, action plans, geological and geotechnical data, and results from the mine – production, costs and coal quality. The key to providing good data to management is to combine the key information into one management system where all the significant facts are available to all. This then allows personnel to incorporate their process into the overall mine objectives, creating the integrated management process.

The keys to good, efficient longwall mining are short face changes, good development performance (good inventory) and maximising shearer cutting. But to what degree do we analyse the data from these activities? The data from a longwall nowadays is well logged; with the information technology available we can tell exactly where the machine is, when it moves and how fast it travels. But do we sit down and analyse how long the bolting cycle takes in the continuous miner section in the same detail?
At Ricall Colliery in the UK, where there are lengthy bolting cycles, the result of high bolt densities required to counter high stresses were monitored by supervisors who included the monitoring data in their shift reports. This was entered into the management computer system and then analysed by the teams. The review process included the involvement of the miners, supervisors and dedicated management. The changes developed by the process resulted in a performance improvement of more than 20 per cent.

Any good management review process is about accountability.

But what is accountability? It is:

- giving an individual responsibility to complete a task or set of tasks to a target within a specific period of time;
- assign the individual control over all the resources required to deliver the outcome; and
- provide objective positive and negative feedback based on achievement of outcome.

To be successful, accountability reviews must include:

- clarity,
- involvement,
- control,
- support, and
- feedback.

Each plan and action will have had its owners who have been involved in the drawing up of the plan. Each task should have a key person responsible for the outcome. Progress is reviewed, and deviations are noted and reviewed with the accountable team member. The review can be either formal or informal; the reviews are a mixture of both.

Developing and achieving an effective review process results in a target led process. Measuring outcomes against the plan is essential. Introducing key performance indicators also improves the accountability. This creates a performance or delivery based culture.

What should be reviewed? Effort should be concentrated on the cyclic events not just focussed on the major stoppages. For example, turnaround times at face ends represent a delay to coal cutting. Reviewing performance of this activity, which occurs many times in the run of a day, can generate significant improvements. Every minute saved here could result in another 15m of coal cut. The impact of thorough review of this simple, repeated activity can actually increase revenue. Lost production from a major breakdown will never be recovered.

Management need to keep track of the key performance indicators. One needs to keep in mind the old saying ‘what you measure is what you get’. Monitor performance, drive improvement and signal the key priorities. The use of a balance scorecard with those key metrics of safety, development inventory and so forth. The key targets: costs, maintenance downtime, productivity improvement, development inventory and so forth. The details would vary from site to mine. From mine manager to shift supervisor the feedback initially was ‘we do this already, why introduce this formal paper system’, ‘waste of money these consultants telling us what we already know and do’, ‘we need to be at the coalface where we make it happen’.

This is a common theme when managing change. The people who ‘the management of change’ will most affect, can’t see what the differences are between the old and the new, or how the change will affect their daily lives, or what changes will actually take place and why there is a need to introduce the change in the first place.

Figure 4 shows a typical ‘bath tub’ diagram commonly used by management theorists when describing the change process. And from the authors own practical experience the model is very close to what occurs in practice.

**MANAGEMENT OF CHANGE**

**Implementation of change**

Enough about the process; what about the management of change and the implementation? When reading this you may be asking yourself ‘what’s so unique and different about this process – we do this at our mine’.

This is exactly the response UK Coal had initially from every mine. From mine manager to shift supervisor the feedback initially was ‘we do this already, why introduce this formal paper system’, ‘waste of money these consultants telling us what we already know and do’, ‘we need to be at the coalface where we make it happen’.

This is a common theme when managing change. The people who ‘the management of change’ will most affect, can’t see what the differences are between the old and the new, or how the change will affect their daily lives, or what changes will actually take place and why there is a need to introduce the change in the first place.

Figure 4 shows a typical ‘bath tub’ diagram commonly used by management theorists when describing the change process. And from the authors own practical experience the model is very close to what occurs in practice.

**Managing the change**

Implementation is about building a team to carry out the change, finding the key enablers for change, setting targets for the change which are aligned to the strategic vision, developing a process to carry out the change, and then ensuring a system is in place to monitor and review the change process (Figure 5).

**Building the change team**

For company-wide transformation change to be successful, the CEO must be seen to lead the change. The CEO can then identify the key positions and assemble the team. This team is then empowered to make the changes with the CEO’s blessing and involvement. The CEO must develop the vision and strategy for change, and his team of champions will make it happen.

**Setting targets**

The implementation plan, for UK Coal, started off by identifying the key targets: costs, maintenance downtime, productivity improvement, development inventory and so forth. The particulars would vary from mine to mine, but what was important was the key focus on what actions were required to
drive the improvements. It is important to establish the importance of metrics, stretch targets, and performance tracking in raising the performance standards of an organisation, as outlined below:

- metrics must be balanced, consistent and linked to strategy:
  - balanced metrics promote comprehensive measurement;
  - ‘what gets measured gets managed’;
- target setting must be customised:
  - high enough to stretch the organisation, realistic enough to motivate action;
  - strategic metrics must be driven down to operational levels:
    - metric cascades facilitate the metric linking process;
    - ‘actionability’ increases as metrics are translated to lower levels;
- accountability should be established for all metrics and associated targets:
  - integration with individual performance incentives is critical.

The target setting was customised and based on:

- historical performance,
- benchmarking across the mines, and
- benchmarking worldwide.

The targets were set to stretch the organisation to meet its potential, representing significant improvement over current performance levels.

For example, each mine was set an operational target for improving its efficiency. Most mines were operating longwalls at 50 per cent or less of full potential (based on theoretical figures on machine speeds and total time available to mine with the longwall shearer). World best practice had some mines over 80 per cent. In the past some mines within UK Coal had operated at 65 per cent to 75 per cent. So mines were given realistic stretch targets based on their own historical records.

**Rolling out the new management process**

Using the planning process the mines had to develop a key action plan on how these targets were meant to be achieved and what was required to be done differently. These plans were then incorporated into the planning process.

One mine was chosen to pilot the planning process. Other mines were involved in developing the new maintenance program, development inventory improvement, and others in methods to improve longwall relocations.

Once each of the initiatives had been piloted, a rollout program was developed that involved training the many personnel through various workshops.

A UK Coal manual was produced on the ‘UK Coal Way of Management’ and each Mine Manager was trained and issued with these at a two day seminar.
The rollout program involved the change team going to two mines simultaneously and rolling out the process through training and hands on assistance in the new planning process. The Group Director and his team were heavily involved in the rollout and were seen to be the endorsing the program. Over a period of 12 months all the mines were operating the system.

A vital factor in any change program is communication. This needs to be frequent and involve the whole workforce, not just the management teams. Throughout the program, the change team used the company’s newsletter as a vehicle to promote the quick wins and keep the employees informed of the many initiatives that were taking place.

Communications were not restricted within the company; presentations were made to city analysts who were frequently updated on progress.

RESULTS

Through this structured change and planning process the capability within the company changed and productivity improvements were realised.

The real change over the two years at the mines included:

- mine manager understanding and leadership;
- new talent introduced from outside the coal industry with new fresh ideas;
- low tolerance for failure and meaningful targets;
- organisation by process;
- problem solving, planning-oriented approach;
- high workforce involvement;
- willingness to deploy best teams to important places;
- execution orientation – ‘just do it’;
- alignment of incentives; and
- cross-mine transfers of best practice.

These changes resulted in the following improvements:

- improved productivity at some mines to over 65 per cent efficiency;
- reduced operating costs by A$ 80 million;
- reduced downtime improved by more effective planned maintenance;
- a clear strategy for closure of mines with low economic valued resource; and
- improved profits in deep mines by 2003 of A$ 38 million.

The UK Coal initiative saw the company’s overall profits increase and in 2003 a small profit was made, which was a significant change to the losses in the previous years. The share price rose to new levels in 2004 on the back of business improvements.

CONCLUSION

Coal mining in the UK has recently involved a great deal of uncertainty. When mining at depths of 600 m to 1000 m the equipment is under greater stresses and the geological risks are higher. To ensure world class performance, world class management techniques are required. UK Coal is still evolving, but with a clearer defined strategy for business improvement, streamlined organisational structure and a more dependable management process, it has the tools to continue to improve performance.
Reducing the Variability in Dragline Operator Performance

G Lumley

ABSTRACT

Australian coal mines have spent considerable capital on dragline improvements over the last two years. This has included ~$30 M on UDD conversions, >$20 M on new buckets, boom upgrades, electrical upgrades, etc. This is a natural part of the response of mining companies to technology and replacement programs. Capital expenditure is a normal part of the ongoing success of most companies. There is however, a tendency for some people to rely on the capital alone to provide the ongoing improvements in equipment productivity. Implementing new technology through capital expenditure is only part of the equation in continuous improvement. For 80 - 90 per cent of the year, the operator controls the productivity achieved by the dragline.

Variation between operators is huge. The average standard deviation in productivity is 12 per cent and maintenance impact is over 40 per cent. Robbins 2003, states:

Contrary to what we were taught in grade school, we weren’t all created equal. Most of us are to the left of the median on some normally distributed ability curve.

Further, he states:
The issue is knowing how people differ in abilities and using that knowledge to increase the likelihood that an employee will perform his or her job well.

There are two options for reducing variability between operators; improving operator ability and getting the machine to take over what the operator is doing (automation). Dragline automation will be discussed, however, this paper will focus more on the ‘human factor’ and how to establish a dragline with minimum variability between operators.

INTRODUCTION

Minor variations in dragline productivity can be leveraged into large variations in coal production and mine profitability, (Hettinger and Lumley, 1999). Given that a one per cent increase in dragline productivity is valued at between $50 000 and $2 300 000 per annum, (GBI Consulting Pty Ltd, 2004), it is not surprising that significant interest has been shown in dragline productivity over the last 20 years. But why do some draglines continue to out-perform others? Why has so much of the research money spent on improving dragline productivity not been reflected in improved productivity? Why, over the last few years, has there been an emphasis on capital improvements rather than the cheaper option of process improvements? Why have we, as an industry, largely left the operators without sufficient support? Peterson, Latourrette and Bartis, 2001, state:

...despite the prospect of automation and other technology enhancements, people are becoming more critical to the success of a mining operation, not less.

This industry can’t afford to be satisfied in the gains achieved over the last ten years. In 2003/2004, the average Australian dragline underperformed best practice by 25 per cent. The average Australian dragline was 46 per cent below ‘best feasible’. Some of this average 46 per cent difference may only be achieved through capital expenditure, eg stronger booms, better motors, high productivity buckets, etc and some of it will never be achieved, eg higher payloads cause slower swinging, faster swinging causes more downtime, etc. However, it is this author’s belief, supported by other industries’ experience, that at least half the difference between current performance and best feasible may be achieved through process improvements. Those process improvements are heavily reliant on human factors.

Peterson, Latourrette and Bartis, 2001, recount a mining executive’s response to the greatest constraint to his organisation improving productivity as ‘Getting people to think!’ Mine site productivity starts at the top and the attitudes and actions responsible for this productivity permeate through the whole workforce. Good attitudes lead to good productivity while poor attitudes lead to poor productivity.

Some sites get caught on the three ‘P’s’; personalities, power and politics. People get in the way of objectivity. More than ever, the need for objectivity in evaluating dragline performance is essential. 19th century American humorist, Artemus Ward, put it very cleverly when he said, ‘It ain’t the things we don’t know that gets us into trouble. It’s the things we know that ain’t so’ (Zikmund, 2003).

WHAT IS DRAGLINE BEST PRACTICE?

Defining ‘best practice’ is not a simple matter. In this paper, best practice is referred to as the average of the top ten per cent of dragline years in the GBI dragline productivity database.

The GBI database contains data from draglines all over the world and, as of April 2005, contains more than 150 million cycles spanning nearly 500 dragline years from Queensland, NSW, USA, South Africa and Canada.

Table 1 summarises the results of an analysis of the productivity of the best performing ten per cent of draglines in the database. It shows the average figures achieved by the top ten per cent of draglines (‘best practice’) against the average of all the draglines in the data with the difference noted in terms of impact on productivity. An average in situ SG of 2.20 t/m³ is assumed.

<table>
<thead>
<tr>
<th>Key performance indicator</th>
<th>Average</th>
<th>Best practice</th>
<th>Impact</th>
</tr>
</thead>
<tbody>
<tr>
<td>Daily swings (#)</td>
<td>861</td>
<td>957</td>
<td>+13.0%</td>
</tr>
<tr>
<td>Payload</td>
<td>88</td>
<td>103.1</td>
<td>+17.2%</td>
</tr>
<tr>
<td>Fill time (secs)</td>
<td>15.8</td>
<td>14.9</td>
<td>+1.6%</td>
</tr>
<tr>
<td>Swing time (secs)</td>
<td>25.3</td>
<td>25.7</td>
<td>-0.5%</td>
</tr>
<tr>
<td>Ret time (secs)</td>
<td>21</td>
<td>20.1</td>
<td>+1.6%</td>
</tr>
<tr>
<td>Spot time (secs)</td>
<td>4.7</td>
<td>3.5</td>
<td>+2.2%</td>
</tr>
<tr>
<td>Cycle time (secs)</td>
<td>66.8</td>
<td>64.2</td>
<td>+4.8%</td>
</tr>
<tr>
<td>Dig time (%)</td>
<td>66.6</td>
<td>72.4</td>
<td>+8.2%</td>
</tr>
<tr>
<td>Productivity (BCM/day)</td>
<td>34.440</td>
<td>44.850</td>
<td>+30.2%</td>
</tr>
<tr>
<td>Productivity (BCM/yr)</td>
<td>12.6 M</td>
<td>16.4 M</td>
<td>+30.2%</td>
</tr>
</tbody>
</table>

The difference between the average of the top ten per cent and the average of the whole database is illustrated in the waterfall chart depicted in Figure 1. The blue bars represent activities where performance is better whilst red bars show areas where the performance is worse.
Of further interest is the trend in dragline performance. Figure 2 shows average dragline productivity from 1997 to 2003. Average dragline productivity has increased by 17 per cent since 1997 while best practice productivity has increased by 10.5 per cent. In both cases the key component is payload which is up by 7.1 per cent on average and 6.9 per cent in best practice draglines. The potential to increase further is also demonstrated on this plot. The most productive dragline, (normalised to this class), is also shown on this plot along with the best feasible performance which is a combination of the best key performance indicators (KPI’s). The average dragline in 2003 needs to improve by 26 per cent to achieve average best practice and 48 per cent to achieve feasible best.

Figures 1 and 2 show that most draglines have huge potential to improve productivity. In 2003/2004, the average Australian dragline underperformed best practice by 25 per cent. The average Australian dragline was 46 per cent below ‘best feasible’. Some of this average 46 per cent difference can only be achieved through capital, eg stronger booms, better motors, high productivity buckets, etc and some of it will never be achieved, eg higher payloads cause slower swinging, faster swinging causes more downtime, etc.

It is this author’s belief that the average Australian dragline has >20 per cent improvement available through process improvements, largely through improving the ‘human factor’. Even those Australian draglines achieving current ‘best practice’ (>17MBCM for a BE1370W/M8050) have >10 per cent improvement available. This corresponds closely to the one standard deviation (12 per cent for draglines) achieved by other industries when the ‘human factor’ is controlled (Newman, 2004).

**WHAT KPI’S ARE CRITICAL IN IMPROVING DRAGLINE PRODUCTIVITY?**

Figure 1 demonstrates that the best practice draglines achieve higher payloads and higher dig hours than the average – thus a concentration on payload and related issues has the greatest potential to increase productivity. To demonstrate the importance of certain KPI’s, the strength of the relationship between the key performance indicators (ie payload, fill time, swing angle, etc) and the productivity, is calculated. The strength of the relationship of each KPI is quantified by the correlation coefficient and is described as the \( r^2 \) value.

\( r^2 \) is the relative predictive power of a model (in this case, the formula of the linear relationship) and is a value between zero and one. The closer it is to one, the stronger the relationship where ‘stronger’ implies a greater ability to predict. This is extremely helpful because it shows which KPI’s have the strongest relationship to productivity. The \( r^2 \) values for the KPI’s relative to productivity for nearly 500 years of dragline data are:

- Payload: \( r^2 = 0.92 \)
- Dig time: \( r^2 = 0.40 \)
- Cycle time: \( r^2 = 0.08 \)
- Fill time: \( r^2 = 0.00 \)
- Swing time: \( r^2 = 0.01 \)
- Swing angle: \( r^2 = 0.02 \)
- Return time: \( r^2 = 0.01 \)
- Spot time: \( r^2 = 0.30 \)

It is clear that there is a strong correlation between productivity and payload whilst fill time has a negligible relationship with productivity. This confirms that the most productive draglines achieve high payloads – even at the expense of fill time and other components of the cycle time.

**WHAT OPTIONS ARE AVAILABLE TO REDUCE DRAGLINE OPERATOR VARIABILITY?**

Dragline automation

To determine the desirability of automating parts of the dragline process, individual components of the cycle are separated and the efficiency of those parts of the operation determined.

**Fill efficiency**

The fill time and fill distance can be used to determine how efficiently the operators have filled the bucket. A plot of fill time versus fill distance cycle by cycle is created. It is normal for this plot to show a vast spread of results above a fairly well defined lower boundary. A sample plot is shown in Figure 3.

Where the bucket was ‘perfectly’ easy to fill the average would fall on the peak performance line. This peak performance line represents those cycles where the operator did everything right and is representative of the peak motor output.
Taking this one step further, it is proposed that as the average moves away from the peak performance line, the more difficulty the operator is having in keeping the bucket travelling at the speed the motors will allow. The filling efficiency is defined as the fill time on the peak performance line for the fill distance achieved divided by the average fill time.

The trend in fill efficiency over time is useful in determining operator filling performance. The factors which may impact on the filling are:

• geology,
• blasting,
• drag motor performance,
• engage location,
• operator ability and performance, and
• bucket behaviour.

**Swing, hoist and return efficiency**

The extended analysis of swing performance versus swing angle produces a swing efficiency factor, which is calculated in the same way the filling efficiency factor was determined. The same can be done for swing time (for hoist dependent cycles) versus hoisting time and return time versus swing angle as shown in Figure 4.

Figure 4 can be created for each time period and the peak performance at the average swing angle divided into the average swing time to give swing efficiency. The factors that may impact on swing time are:

• mine plan,
• swing motors,
• operator ability and performance,
• drag payout, and
• payload.

The standard deviation of fill rate, swing rate and return rate on a cycle by cycle basis are typically 40 per cent, 30 per cent and 40 per cent respectively of the average. A significant part of this variability can be attributed to the operators and more specifically, differences between operators.

The major attempt, which the Australian coal industry has made in this area, has been work at the CSIRO over the last 12 years on automating the swing, dump and return parts of the cycle. The latest step was a trial of the system on the BE1350W.
dragline at Boundary Hill (CSIRO, 2003). The system is able to match or exceed operator performance in some, but not all, cycles. This system proved the following points:

1. return time was significantly better than swing time;
2. skills such as bucket disengage, dumping and recovery are performed consistently well;
3. the system was highly reliable; and
4. the system’s interface was intuitive and readily accepted by the operators.

The report explains that the computer’s skills are not perfect but the trial demonstrated that they may be improved with further work. The lack of terrain data caused the system to require larger margins of safety in bucket trajectory than what the operator may use. Consequently, the two key areas of future work required are the integration of a form of machine vision (which CSIRO is working on) and the refinement of the swing and return algorithms.

Right now, the CSIRO – Dragline Swing Assist system is not ready for commercial release and the timing for it becoming available is not possible to predict.

**Operator impact**

During the 1980s and 1990s an industry-wide culture of industrial deadlock and regulatory institutions that quarantined Australian coal operations from global competitive pressures, made workplace reform very difficult, (Goldberg, 2003). The wealth generated from coal operations provided relatively little for the shareholders. While the rest of the Australian mining industry responded to the opportunities and threats of globalisation, the nation’s coal sector didn’t. In terms of safety, productivity and profitability, coal operations were increasingly out of step.

The possibility that the industry would lose considerable market share to competitors in Indonesia, South Africa or Central America appeared very real as little as seven or eight years ago. The performance especially in Australia, especially in NSW, was ‘abysmal’. However, some new ventures were doing things differently and securing better outcomes, (Davies, 2001). During one six week strike the staff ran the operation. The performance – admittedly under abnormal circumstances – demonstrated the efficiencies that a more flexible operation could achieve.

Thanks to the changes that have taken place in the workplace, there has been a significant improvement in productivity and an accompanying reduction in costs. Figure 5 shows the improvement in productivity through the late 1990s.

Many of the structural changes and improved work practices sought in 1997 have been achieved (Goldberg, 2003). Australian coal mines in general have become more efficient and more profitable. A major part of this is the ability of the mines to choose the employees they want to hire based on merit rather than seniority. Davies (2001) states that one of the major improvements was the ability to make changes without having to first ask permission from the union. Retention of the ‘best workers’ at a mine in late 1998 was the first time a merit based selection process had been implemented for retrenchment in the NSW Coal Industry.

There has been little documented on the variations between dragline operators apart from private work undertaken by this author and his company GBI Consulting Pty Ltd. The average difference between best and worst operators is 35 per cent in productivity and 140 per cent in damage impact (Lumley, 2004). Standard deviations are 12 per cent and 40 per cent respectively. An example of this is Figure 6 which shows a month of dragline data reporting each operator on a plot of productivity versus damage.

The plot shows a month of data where the most productive operator (5) was 33 per cent more productive than the least productive (14). The most damaging operator (13) caused 150 per cent more damage than the least damaging (3). This is a dragline that consistently achieves productivity in the top ten per cent of draglines worldwide and would be considered above average in terms of ability of its operator teams. All of GBI’s private work would suggest that this large variance is the norm (or better than the norm) rather than the exception (Lumley, 2004).

Unfortunately, as a general statement, this industry has not supported the operators well. This is demonstrated clearly by the variation in performance. Consider the following Australian statistics:

- the variation in productivity between operators is significant;
- 35 per cent average difference between most and least productive; and
- 12 per cent standard deviation;
- the variation in maintenance impact between dragline operators is extremely high;
- 140 per cent average difference between least and most ‘damaging’; and
- 41 per cent standard deviation.

For draglines which have operated over 20 years with very low turnover of operators, why does this scenario of large variations between operators occur?

The large variation is a function of several inter-related factors:

- Poor management practices/attitudes.
- The historical system of equipment operators being chosen based on seniority; that is, the longest serving employee, who wanted the available job, got it.
- A shortage of trained operators stemming from the following:
  - Training for operators has traditionally been done ‘on the job’. Hence, a trainee learns by doing, with the resultant exposure for the mine to reduced safety, increased damage and reduced productivity. On hugely expensive equipment training was a large cost and risk factor. The logical consequence was that training was done on an ‘as needed’ basis and generally, only the minimum number of people were trained.
  - The industry is expanding. New mines are opening, eg Coppabella, Hail Creek and Rolleston. Other mines are expanding, eg Newlands, Blackwater, Goonyella/Riverside, etc.
  - Mine workforces are ageing; particularly in some of the areas of low turnover, ie dragline and shovel operators. The natural consequence has been an acceleration of trained and experienced people retiring and leaving the workforce.

The logical consequences of these factors are:

- many current operators doing jobs they are not naturally suited to;
- no pool of trained operators to replace the under-performing operators; and
- very large variations in operating performance on very expensive pieces of equipment.

For 80 - 90 per cent of the year, the operator is in control of the productivity achieved by the dragline. Therefore, the greatest potential for gains to be achieved by the dragline is by providing support to the operator.

Support for the operator may be in the form of:

- management approach,
- ensuring they are naturally suited to the job before they start,
- training, and
- setting targets and performance feedback.

These points are addressed in turn.
Management approach

Productivity is about attitude. Much can be learnt about the theory behind operating draglines and improving productivity but if the mine does not have a ‘culture of productivity’ then achieving best practice is virtually impossible. The productivity attitude must be established and supported from the highest level on the mine site. The experience at Robe River mine in 1986 (Copeman, 1987), provides an excellent account of the way companies may act if they are not happy with mine site attitude. It is important to note who were the first people dismissed at Robe River.

Profitability is usually highly leveraged against the productivity of the dragline and thus significant management effort and enthusiasm should be focussed on getting the most out of the dragline. Exactly what this entails is not always well understood and often other activities are given preference sometimes to the detriment of dragline productivity. The actions of mine planning, blasting, scheduling, maintenance and man management all play a significant role in production but need to have a common productivity focus or else they can negatively impact the dragline performance as shown in Figure 7.

In Figure 8 the productive dragline shows a different flow of ‘impacts’. The dragline productivity is made central to the mine’s performance. People and personalities become less important and the dragline productivity becomes of primary importance.

The dragline productivity now ‘drives’ other aspects of the mine operation. It is no longer acceptable for other people to impact productivity negatively; they know what is expected of the dragline and they should do their job in such a way as to help the dragline achieve it. This is the way which all mines achieving best practice operate.
Tests of maximum performance using test tasks analogous to the duties of the job are used to identify the abilities of a prospective employee. However, such ability tests typically do not measure important personal style characteristics such as honesty, reliability, leadership style, job involvement, customer service orientation, team spirit, etc. Further they do a less than perfect job of measuring the elusive variable known as common sense which is important in the workplace (Bersoff, 1988). The best estimates suggest that ability test results by themselves probably account for about 25 per cent of the variance in job performance (Angus, 1996).

Numerous studies have shown that modern psychological testing is one of the most valid predictors of future job performance. With increasing frequency, employers are now turning to testing to aid in selection decisions as well as evaluation of personnel.

Comparisons of human attributes and differences have a very long history (Fröschl, 2001):

- Hippocrates – (400 BC) attempted to theoretically define four basic temperament types: sanguine (optimistic), melancholic (depressed), choleric (irritable) and phlegmatic (listless and sluggish).
- Galton – (19th century) measured human individual differences in terms of ability to discriminate between stimuli.
- Binet – devised tests to measure differences in specific human abilities. Now numerous tests measure specific abilities, strengths and competencies.
- 1917 – Psycho-Technical Test Office opened in Germany.
- Army Alpha and Beta tests (WW1 and WW2) – developed out of an urgent need to select personnel with specific aptitudes for training in specialist and strategic roles.
- 1990s – Computer-aided test procedures developed.
- 2000s – Linkage to statistical packages such as SPSS.

The reasons for using testing have not changed over time. They are rooted in the necessity to place the right person in the right job. Significant studies have been conducted on using psychomotor testing in several industries including aviation, train operation and driving. Previous work also demonstrated that the computer based ability testing was a predictor of student naval aviator and naval flight officer performance (Delaney, 1992; Street, Chapman and Helton, 1993). Portman-Tiller, Biggerstaff and Blower (undated), report on testing conducted by the Naval Aerospace Medical Research Laboratory. They found that the historic paper-based system and a computer based system have some commonalities but the computer based system offers significant advantages in aviation selection. Specific tests in tracking and dichotic listening tasks are similar to the real world environment and provide predictive validity using a flight performance criterion.

Schuhfried, in Traffic Psychology Psychomotor Testing Research Report (undated), reports that psychometric testing and psychological performance is an excellent predictor of future driving performance. Strong correlation is found between driving performance and tests for concentration, visual perception, reaction speed and intelligence.

History and results from the Psycho-Technical Test Office of the Deutsche Reichsbahn, (German Railways) indicate that the system is able to provide significant information as to the suitability of applicants for a range of jobs (Fröschl, 2001).

Various mines have used psychomotor testing to select equipment operators, including Coppabella, Belgium, Hanley, Ekati, New Mexico, Syferfontein and Kwagga. The results from these mines have not been published.

A strong correlation was found between computer based test results and a qualitative assessment of 28 BHP dragline operators (Moore, 1998).
As can be seen from Figure 9, the results were predictive of operator performance (statistical regression results $r^2 = 0.78$), suggesting that it is likely to be effective as a selection device. However, the reliability of the results is limited by the size of the sample and the qualitative approach used for field assessment. From a research perspective, the sample is of doubtful value because it is unclear how the sample was selected (ie potential sampling bias). A sample of 28 in an industry population of about 780 provides a confidence interval of 18.9 per cent (ie potential statistical power issue). This means the industry result extrapolated from this data could be plus or minus 18.9 per cent. This doesn’t change the trend identified in the data, but it makes it virtually impossible to place an accurate cut-off between acceptable and not acceptable performance.

It has been demonstrated in several industries that people have different cognitive abilities. Put simply, everyone is different. Different people are suited to different activities. The coal industry endured significant pain through the mid to late 1990s and early 2000s as companies sought reforms to work and management practices. One key issue to come from this ‘pain’ is that most mines’ selection processes are no longer constrained by considerations of seniority. Most mines are now able to select the people they want.

*This is a tremendous opportunity for the coal industry.*

Some mines have put processes in place for selection of suitable people and the aim of this paper is not to undermine any of that work. The position of dragline operating is seen as being a different consideration to other pieces of equipment. The coal industry endured significant pain through the mid to late 1990s and early 2000s as companies sought reforms to work and management practices. One key issue to come from this ‘pain’ is that most mines’ selection processes are no longer constrained by considerations of seniority. Most mines are now able to select the people they want.

A tool has been developed by an Austrian company, Schuhfried, called the Vienna Test System (VTS), which is a computer based testing program consisting of six tests. These tests assess the co-ordination abilities required for safe and productive performance of machinery and vehicle operators.

ACARP is funding a comprehensive program which aims to confirm what combination of VTS output and demographic/environmental/psychological factors assessment is most appropriate for predicting dragline operator performance.

Table 2 shows the factors which the VTS can quantify.

The selection of suitable operators is an area of great potential cost-benefit in the dragline operation. The cost in production by training on the full size machine is between 100 000 and 150 000 BCMs over the first six months. If the trainee is cut after two months the cost is probably about 50 000 BCM, which is worth between $20 000 and $40 000 per trainee. If the VTS (and other factors) works it will cost less than $500 to come to the same conclusion.

**Operator training**

The cost of training can be substantial. Figure 10 shows the performance of a ‘green’ operator over a period of six months on the dragline. In six months, 120 000 BCM productivity was lost.
The challenge for initial training is how to reduce this cost. The challenge for experienced operators is assisting the process of continuous improvement.

Operator training can take several forms:
1. use of a physical or computer simulator,
2. external operator trainer works on the dragline with the operator,
3. internal trainers, and
4. self training.

Each of these has an important part to play in the optimisation of dragline productivity.

By far the most important is self training. This comes from the management attitude and the selection of the ‘right’ people. Self training goes on as long as the person is operating the dragline. The best aspect to self-training is that it costs very little.

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Setting targets and performance feedback

The author’s work has shown that at least 95 per cent of dragline operators are interested in doing a good job. The easiest way to support this is to provide ongoing feedback to the operators. Some mines haven’t progressed to the point where the operators feel comfortable with individual reports. In that case, provide feedback for the dragline as a whole. When starting out, any reporting is better than nothing. Figure 11 shows a simple operator report. The key to operator feedback, regardless of the level of detail, is that it must be discussed with the operator to ensure they understand what the report means. This links closely with the concept of self training. Operators must be encouraged to do better and use the information contained in reports to improve their own performance. Unfortunately, most reporting to operators is wasted because mines do not make the investment in time required to discuss the results with the operator.

CONCLUSIONS

This paper provides guidance on where this author believes the most cost-beneficial improvements in the dragline operation may be obtained. The following are the main points.

- The potential for dragline automation is large but the technology is still being worked on. The CSIRO – DSA system is the first effort to automate part of the dragline operation. It has demonstrated that computer control of a dragline is possible and that it can perform better than a human average in some situations. Enhancements are currently being worked on to overcome the suboptimal performance in certain situations.
- It is this author’s belief that the average Australian dragline has >20 per cent improvement available through process improvements, largely through improving the ‘human factor’. Even those Australian draglines achieving current ‘best practice’ (>17 MBCM for a BE1370W/M8050) have >10 per cent improvement available. This corresponds closely to the one standard deviation (12 per cent for draglines) achieved by other industries when the ‘human factor’ is controlled (Newman, 2004).
- Attitude is important when approaching productivity. If current results are viewed negatively and mines make excuses for why results came out the way they did then nothing will change and productivity won’t change.
- Establish selection processes which target employees who are naturally suited to dragline operating.
- Support training, both external and internal.
- Provide reporting structures for operators.

This author does not advocate the need to spend significant capital to achieve best practice. Whether a mine spends capital on their dragline is irrelevant to the message of this paper. All draglines currently have the potential to improve their processes. The majority of the gains are available through changes in the ‘human factor’. Improving processes; including management, operational issues, maintenance, etc are issues that don’t need large sums of money spent on them.
Application of Tagging Systems for Personnel and Vehicle Access Control

L Marlborough1, S Barrow2 and D Kent3

ABSTRACT

Controlling the access of equipment, and people, has become more critical over the last few years for improving productivity and to meet safety responsibilities imposed by legislated obligations to have safety management systems. Hence Mine Site Technologies (MST) has spent the last three years completing the development of the tagging system to meet the mining industry’s requirements, particularly streamlining access control. The basic tagging system tracks active Tags carried by personnel, or attached to vehicles and equipment, within underground zones.

These Tags are detected (read) by Beacons placed strategically throughout the mine. As a Tag, or Tags, go past a Beacon they are read and the information transmitted back to the main database PC. This allows the current whereabouts of personnel and equipment to be known, as well as the history of their movements.

The objectives of implementing the TRACKER Tagging System at Oaky No 1 Coal Mine were to:

- maintain a count of personnel inbye a particular point, and can alarm when a pre-set number is reached (eg the limit imposed by availability of SCSR’s, etc);
- maintain a count of vehicles (and vehicle type) inbye a point, and can alarm if when a pre-set number is reached (eg when rated diesel units exceeds the limit for the ventilation present in that zone);
- manage the vehicle fleet, by keeping track of various vehicles underground better control and dispatch management of the equipment is achieved;
- keep track of equipment, knowing the location of equipment (such as forks, baskets, pumps, fans, etc) where last used or stored will minimise the risk of losing gear and assist in keeping inventory to a minimum; and
- locate personnel in emergency situations, by placing read Beacons throughout the mine the location of all personnel can be known in real time so that in the event of an emergency, their withdrawal can be monitored.

INTRODUCTION

The release of the TRACKER Tagging System has been timely as controlling the access of equipment, and people, in areas of the mine has become more critical over the last few years. In particular, its application in improving productivity and to meet safety responsibilities imposed by legislated obligations to have safety management systems (the commonly termed ‘duty of care’ principle).

A review by MST (Internal Market Surveys 2000 and 2002; Allen et al, 2002; Einicke, 2003) revealed that none of the tagging systems tried have operated reliably. Because they saw a market opportunity, MST has spent the last three years completing the further development of the TRACKER Tagging System to meet the mines’ requirements, particularly streamlining access control.

THE TAGGING TECHNOLOGY

The system tracks active Tags carried by personnel, or attached to vehicles and equipment, within zones underground. These Tags are basically small radio transmitters that transmit to read Beacons placed strategically throughout the mine.

As a Tag or Tags go past a Beacon they are read and the information transmitted back to the main database PC. This allows the current whereabouts of personnel and equipment to be known, as well as the history of their movements.

However, the key to TRACKER’s development has been its development as a true system. There are a myriad of ‘off-the-shelf’ Tags and readers available, but to operate successfully the system must be truly integrated to suit the underground environment. Importantly this is not just obvious physical requirements (water proof, rugged, etc) but the flexibility to integrated simply into a mine’s existing infrastructure, as detailed further in the discussion on the read Beacons below.

TRACKER is designed for both coal and metalliferous mines, though the actual application varies between the mines (eg metal mines use TRACKER for blast time safety as well as fleet management).

In more detail, the main system components are as follows.

TRACKER Tags

TRACKER Tags transmit their unique ID on UHF frequency to a range of up to 100 metres in a typical underground roadway (50 m inbye, 50 m outbye). They use a tuneable matched loop antenna printed on the circuit board.

The Tags transmit three key pieces of data:

- unique ID,
- battery status, and
- checksum.

Tags are designed to be carried by personnel, or mounted on vehicles and other equipment. Two versions of Tags are used (see Figures 1 and 2).

The first Tag is a small independent (self contained) unit that may be worn on the belt, or attached to equipment using a custom housing. This Tag contains a 4.5 V battery pack with a lifespan in excess of ten to 12 months. The self contained Tag also has an LED to indicate battery status, the LED has three states:

- off = OK,
- blinking = transition indicating approximately one month left, and
- solid = replace.

Low battery status is recorded and displayed by the TRACKER Tagging System (stored in the database and viewable at the Client/Operator’s software).

The second Tag version is incorporated into newer technology cap lamps.

The Tags are currently approved intrinsically safe for coal mine use in Australia, China and the USA.
Tracker read Beacons

Beacons receive Tag ID’s, transmitted by UHF signal, and then transmit that data back to the PC at central office via RS485 serial protocol. To allow flexibility to fit into a mine’s existing communication infrastructure the data links between Beacons and Beacons and the main database PC can be conventional wiring, fibre optic and radio modem links that may all be integrated to carry both up-stream and down-stream data as required. The system can operate over leaky feeder radio systems, but is not recommended as the low data rate of leaky feeder does slow the system response time down and limit the number of Beacons that can be installed.

The Beacon enclosure (see Figure 3) is made of stainless steel and operates from a 12 - 28 VDC supply. TRACKER Beacons can, as a minimum, reliably record ten people, with Tags, moving past in a vehicle at 35 - 40 km/h, and more moving at lesser speeds.

Some key aspects of the beacon’s operation include:

- Beacon typically detects Tag signals via a stubby quarter wave monopole coax antenna;
- Beacons sensitivity can be manipulated to suit the environment (sometimes a short read range is required to ensure a Tag is not read at two Beacons at one time);
- Beacons operate on a board switch mode supply and require DC >12 to <30 volts for reliable operation;
- power consumption is approximately 800 mW (27 mA at 30 volts/70 mA at 12 volts);
- Beacons have a direct read point option, indicating Tag IDs (and Beacon name) as they are detected; and
- three LED indicators display operational status:
  - flashing red LED represents the heartbeat (one flash per second) and indicates the presence of oscillations of the crystal oscillator on the beacon’s PCB;
  - flashing yellow LED represents a Tag read, indicating that the Beacon has received a signal from a Tag; and
  - flashing green LED for communication indicates that the beacon’s stored data has been transmitted to the central office on its request.

The TRACKER Beacon has two main electronics circuits that can be configured to give a range of applications within the system, these include:

- a simple termination point,
- a branch/split point for system expansion,
- a converter – 232 to 485, 485 to optics, optics back to 485, etc,
- an isolated 485 repeater/booster,
- a read Beacon,
- an isolated stub driver, and
- a Beacon with any of the combinations above.

Tracker software

The main TRACKER software is typically located on a PC in the communications or control room on the surface, with networking to other PC’s on the surface and underground as required. At set intervals the TRACKER software sends out a request to each Beacon, asking for its stored list of Tag IDs. After a successful transmit Beacons then have their list cleared and begin to collect the next list of Tags.

The time between polling is set within software, it is typically anywhere between three and 30 seconds, depending on system
configuration. Beacons that see a lot of activity require a faster refresh, where as low activity Beacons need not have their lists cleared so often.

Tag IDs are stored in the TRACKER database, recording which Beacons the Tags have passed, and at what time. The TRACKER software keeps a continually updated record of Tags logged in and out of the system.

Beacon configurations, access limit numbers, zone display text (see next section), etc can all be adjusted from the main PC.

The Open Database (ODBC) compliant TRACKER database allows it to share data with other applications. For example, a future generation of PED software may be able to send a message to all personnel between Beacons 5 and 8 by drawing this data from the TRACKER database, or management tools could draw real time locations of vehicles and personnel to manage these resources.

### Zone Display Units

A fourth component to the system used to increase functionality is the Zone Display Unit. Apart from the safety and management uses of the information TRACKER provides, TRACKER can also be interfaced to large display units (see Figure 4) to manage access into certain areas. Key areas of use in coal mines include:

- **Maintain a count of personnel inbye a particular point:** and can alarm when a pre-set number is reached (eg the limit imposed by availability of SCSR’s, etc). This is aided by the use of large display units that show a constant count of personnel numbers inbye that point. The limit number for personnel entering a particular zone can be adjusted from the main PC if circumstances change (eg more SCSR’s are installed).

- **Maintain a count of vehicles (and vehicle type) inbye a point:** and can alarm if when a pre-set number is reached (eg when rated diesel units exceeds the limit for the ventilation present in that zone). Again, this can be aided by the use of the large display units, which display the diesel unit count and then display ‘stop’ when the limit is reached. The limit number of diesel units entering a particular zone can be adjusted from the main PC if circumstances change (eg ventilation volumes are changed into the zone).

- **General vehicle fleet management:** by keeping track of various vehicles underground allows better control and dispatch management of the equipment as outlined by Gauci, 2004. Improved efficiency in this area is often the main cost benefit justification when assessing the use of TRACKER at a mine. The Zone Display Units can be placed at strategic locations to update drivers of transport road conditions (eg used as an automatic block light system), or urgent deployment to another area.

### IMPLEMENTATION AT OAKY NO 1 COAL MINE

Xstrata’s Oaky No 1 Coal Mine has long been one of the most productive longwall mines in Australia. As such, management at the mine is constantly reviewing technologies and processes to further increase productivity.

Hand in hand with this constant development of productivity has been an emphasis on safety and safety management. In particular, the recent Level 1 emergency exercise at Oaky No 1 highlighted areas where communication and knowing the whereabouts of all personnel during an evacuation can greatly assist the effectiveness of an evacuation and allocation of rescue resources.

With these key drivers, MST is working with Oaky No 1 to implement a tagging system in two stages.

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**Stage 1**

The main driver here is cost reduction and production advantages through better management of vehicles and, in particular, implements (such as buckets, forks, pumps, augers, etc). The implementation of a tagging system to control these assets not only gives immediate and quantifiable cost benefits, but provides a deeper understanding of the tagging system’s working parameters so its expansion into a full blown safety system is better understood and managed.

Stage 1 involves installing read Beacons at key travel road locations to log the movements of vehicles, and what implements are attached to each vehicle, as they travels around the mine. Each vehicle and implement will be assigned a unique Tag that is then associated with them as they move around the mine. To achieve this 18 Beacons are installed at strategic points as shown on the mine plan in Figure 5.

This layout shown in Figure 5 provides sufficient size zones to effectively track the movement of equipment within key areas underground.

Key benefits identified in Stage 1 include:

- Knowing the location of all implements will eliminate, or greatly reduce, the time personnel spend searching for a particular vehicle or implement.
- Better management of contractor’s resources on site (what goes in, come out).
- Items like pumps that are move or changed out, are tracked to ensure the correct pump has been moved, and moved to the correct location.
- Access control of vehicles, as each vehicle to be uniquely identified, the system will streamline access control for diesels into ventilation zones, as mentioned in the section on Zone Display Units previously. Each ventilation zone has a limit for diesel horse power depending on the ventilation volumes in that particular zone. As each vehicle type has a rating (tokens), dependent on horsepower, being able to differentiate individual machines ensures the correct horse power count is maintained in each area.

### Stage 2

The system is expanded to 35 to 40 Beacons and all personnel that enter the mine are equipped with a Tag. Each Tag is a personal item and the ID is associated with that person only, and is matched with their cap lamp number.

To complement the basic read functions of some Beacons the Zone Display Units used for diesel access control, will also be used for personnel access control. These Zone Display Units provide information locally as it is processed by the main TRACKER PC in the surface control room. The Beacon layout for Stage 2 is shown in Figure 6.

The key drivers for the implementation of Stage 2 include:

- Access control of personnel; similar to the logic behind vehicle access control, personnel access can be controlled in relation to the available long duration SCSR’s in the panel.

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FIG 5 - Stage 1 Beacon locations.

FIG 6 - Stage 2 Beacon locations.
being entered. The same Zone Display Units that already display diesel limit information now also display a count of personnel logged inbye, and alert the person about to enter if they (or them) will exceed the limit for that zone.

- Should an emergency situation arise and an evacuation instigated, the withdrawal of people from the mine can be monitored to ensure all personnel are exiting, or to highlight any area where personnel are not withdrawing.
- During the evacuation, personnel may not be aware of the availability of equipment (eg a PJB) to assist their exit. Knowing the location of a tagged diesel means that personnel withdrawing form a certain area can be alerted to the location of the PJB closest to them.

**CONCLUSION**

The implementation of the TRACKER Tagging System at Oaky No 1 has provided a powerful management and safety tool at the mine. However, it is important to note, that it is the integration of the tagging system into the mine’s infrastructure and complementation to other management systems that allow its benefits to be optimised.

For example, during an evacuation the tagging system does provide critical information, but it is the ability of the emergency team to react to and use this information that realises the benefits. Hence, for example, communication back to personnel is vital to alert them of correct evacuation routes, or availability of transport, etc. Systems such as telephones, PED pagers, etc work together to ensure this flow of information is fed back to where it needs to be acted on, as emphasised in MSHA reports, 1998 and 2000; MSHA ETS, 2002.

Another example is that the read Beacons are not intrinsically safe, and as such to maintain there operation during an evacuation, where the incident has resulted in a disruption to the ventilation (eg the main fan is off, or one part of the mine has lost stopping, etc) than the ability to monitor gas levels throughout the mine allows power to remain on for some time after an incident to maintain the flow of information as long as possible.

In summary, the implementation of the tagging system at Oaky No 1 gives mine management a powerful tool that provides immediate benefits to the operation. More importantly, the tagging system allows further benefits to be gained from previous investment in a range of technologies (eg communication and gas monitoring systems) by extending the effectiveness and applications of these installed technologies.

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Understanding Organisational and Personal Behaviours to Sustain High Productivity and Safety

M Roberts

ABSTRACT

Although Australia’s coal mining industry is currently experiencing a pleasing period of price relief, long-term trends toward lower real prices will surely continue. Higher prices attract expansion by existing operators and market entry by new operators. History clearly demonstrates the relentless long-term march toward ever lower real prices.

Therefore, the need for economic returns applies constant pressure to improve productivity. To be sustained, high productivity requires a systematic response based on a solid understanding of critical productivity determinants.

The most powerful determinant of productivity is now widely understood to be workplace culture. Culture largely comprises a combination of behaviours, symbols and attitudes. This paper will examine and review science’s latest discoveries in human behaviour. It will provide a proven, practical framework for understanding behaviour and applying this knowledge.

Significantly, it will highlight why conventional approaches toward improving productivity often provide disappointing economic returns.

The modern understanding of behaviour factors will cover coal face operations. More importantly, because executive and managerial levels in coal mining have significantly greater impact on industry and mine productivity the paper will concentrate on leadership.

Particular emphasis will be placed on use of modern systems and in particular use of what are by far the most powerful drivers of behaviours – measurement, analysis and reporting systems. The paper and presentation will show how conventional systems rooted in antiquated American legislation from the early 1900s stifle productivity improvement and drive suboptimal and even counterproductive behaviours. These commonly used systems will be compared with accurate and effective modern methods for assessing performance to improve productivity. These principles are proven in all industries and effective application in mining will be highlighted.

Actual results will be used to demonstrate practical approaches successfully applied in both metal and coal mining in Australia and overseas.

PRODUCTIVITY AND SAFETY – CONSCIOUSNESS, LEADERSHIP AND ACCOUNTABILITY

Sustainable high productivity is the key to commercial survival

Although Australia’s coal mining industry currently enjoys a period of price relief, long-term trends toward lower real prices will surely continue. Higher prices attract expansion by existing operators and market entry by new operators. History clearly demonstrates the relentless long-term march toward ever lower real prices.

The coal industry is not alone. As the world shrinks with more competitive transport and rapid technology transfer in all industries, global markets are increasingly competitive. In the automobile industry, major international markets are enjoying record sales, yet 40 per cent of car-making capacity is idle. Only Honda and Toyota are making adequate returns on investment.

The law of supply and demand and many decades of history show it is to this situation the coal industry will inevitably return.

Only producers with sustainable superior productivity will produce returns commensurate with the cost and risk of investment.

The need for economic returns applies constant pressure to improve productivity. To be sustained, high productivity requires a systematic response based on a solid understanding of critical productivity determinants.

Improving productivity involves understanding and improving the efficiency of core business processes and understanding and improving human behaviour. The latter is particularly important in mining since labour costs can be up to 50 per cent of operating costs.

Even in mines with proportionately low labour costs, productivity is determined by the efficiency and effectiveness in using human physical, mental and emotional energy. A significant amount of this energy is wasted or misdirected or at best not managed optimally. This clearly represents a huge opportunity for improving productivity and safety and for influencing education of future executives and managers.

The most powerful determinant of productivity and safety – organisational culture

Indeed, the most powerful determinant of productivity is now widely understood in all industries to be workplace culture. Culture is essentially the combination of behaviours and attitudes, ie what people do and how they feel toward what they do.

Please note that for the purposes of this paper, behaviour is defined as the observable manifestation of human physical, emotional and mental energy.

Consciousness and the use of human energy – accountability

Effective and efficient use of human energy and time requires all people to make clear, conscious choices in using their energy and when managing other people’s energy. Consciousness, is the key to productivity and safety.

The ability to proactively make conscious choices also affects levels of accountability.

Consciousness and leadership

Leadership is exercised by influencing people to choose to use their time and energy productively to achieve a shared purpose. Effective leadership is about the conscious, committed use of human energy and time. It is about the way people choose to use these precious resources. It’s about choices. It’s about a high level of conscious awareness.

Therefore, this paper asserts that effective leadership starts with a solid understanding of human behaviour and organisational performance. The task of leaders is to apply that understanding effectively to inspire and lift people’s vision, standards and performance. The paper will provide a proven, practical foundation for providing effective leadership and successfully managing change through understanding science’s latest discoveries in human behaviour combined with age-old wisdom readily available to all.

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THE CHALLENGE TO IMPROVE SAFETY AND PRODUCTIVITY

Australian mining has a justifiably proud record of technological prowess. Nonetheless, many mines are not performing to their expected capability. In attempting to address this, managers often initially attempt technological and/or commercial solutions. Experience and success in mining and in other industries revealed the need during the 1980s to address cultural issues and better manage the human resource. It is against this background that a broad overview of traditional approaches to productivity improvement is now being reconsidered.

Hiding behind delusion of increased capital investment, new technology and blind cost cutting

Conventional approaches to productivity improvement fail in three main ways. The first two involve hiding behind capital investment or new technology as a panacea to increase theoretical production capacity. In the 1980s General Motors (GM) embarked on a 21 billion dollar automation campaign to increase productivity. Due to lack of understanding of core assembly line processes this project resulted in robots welding other robots. GM’s Japanese competitors though were using older technology yet had superior understanding of core processes via simple process based measurement, analysis and reporting systems. As a result they continued to outperform GM.

Thirdly, cost cutting by blindly reducing head count has been counterproductive when processes are not understood. In the 1990s Australia’s then largest underground mine reduced head count by 30 per cent for the directors to proudly proclaim annual savings of 100 million dollars. Just two years later the site’s new executive general manager reportedly stated that for each dollar saved in cost cutting, it was costing two dollars due to lack of maintenance and curtailed mine development.

Clearly, wise capital investment, effective new technology and productive cost-effectiveness campaigns can work. It is obvious though that to improve efficiency of resource usage, managers need to improve core processes. Yet conventional accounting systems prevent understanding of processes. Capital investment can then be inefficient or even squandered. Cost cutting can reduce cost-effectiveness. Kaplan and Cooper (1998) and a growing number of authors provide ample evidence supporting this conclusion. Worse, conventional accounting systems drive counterproductive executive and managerial behaviour.

Detailed attempts to improve organisational culture

Another approach is the attempt to change organisational culture. When attempting to lead organisational change, executives and managers are often exhorted by consultants, authors, human resources advisers and/or psychologists to embark on extensive campaigns to change people’s attitudes in the belief that attitude change will lead to behaviour change and thereby higher productivity and improved safety. Consequently, in attempts to improve productivity and/or safety, people have been packed off to numerous training courses to develop skills in teamwork, leadership, communication and interpersonal relationships.

Take the example of attempting to develop teamwork in an organisation in which formal and informal systems measure and reward individual performance. Once executives and managers realise teamwork may potentially be more productive they can choose to send people off on courses in which participants are cajoled or threaded through rope nets, abseil down cliffs or paddle down white-water rapids. Yet back in the workplace, systems remain the same and still measure and reward individual performance to drive individual behaviour. Managers then wonder why such courses actually fail to develop teamwork. They face disappointed employees who understand the projected benefits of teamwork yet still misunderstand how ingrained and habitual systems drive individual behaviour.

The point is that if the systems are not changed, even the most effective training and communication will not produce a change in behaviour. Pre-existing behaviours continue. The underlying assumption that enhanced training and communication will then change attitudes which will in turn then change behaviours is flawed. Worse still, it is sometimes based on the invalid assumption that it is possible to change people.

Many adventures in managing organisational change have fallen dismally short of expectations. This, however, does not mean people are not the key to profitability and safety. To the contrary, even unsuccessful examples of managing change have confirmed the importance of people. The conclusion is that in order to be successful, leaders in Australian mining need to concretely understand human behaviour and process improvement.

Conventional approaches to improving productivity rely simply on doing more of the same; working harder and improving technical skills and knowledge. There’s more to improving productivity than cutting costs and increasing capital, physical effort, skills and technology. Bringing in more production resources, spending more capital, working harder, increasing training and boosting communicating at best have limited impact on productivity and at worst can reduce productivity.

Systems driving counterproductive behaviour

In this context a system is identified as a collection of actions or elements that provide a specific method, procedure or guide. A system is simply anything that drives ways of doing things.

Commonly in organisations, some formal systems may have been built consciously. Others may have been built deliberately yet unconsciously. Many systems may be informal and even undefined. In many organisations these have evolved unconsciously in an ad hoc way.

Systems that can drive counterproductive behaviour and/or lead to compromising safety include: conventional measurement, analysis and reporting of performance; conventional performance appraisal systems; poorly designed 360 degree feedback systems; lack of process and behavioural standards; ineffective communication and planning systems; organisation structures cutting across processes preventing adequate control and undermining accountability; overly prescriptive or vague roles and responsibilities.

By far the most powerful system for driving behaviour is the measurement, analysis and reporting of performance. In many organisations this system has changed little since those developed in the early 1930s in response to antiquated American legislation based on fear arising out of the Great Depression. This is still the case despite significant changes in business and society in the last 70 years. As a result reactive managers use their limited valuable energy to unconsciously stifle other people’s energy or managers waste energy chasing their own tails, or managers give up and succumb to apathy.

Organisation structures are often based on systems developed during the industrial revolution. These were in turn based on earlier systems used by the early military and early church. Such systems essentially go back 2000 years to the days of the Roman Empire and the barbaric values of the ancient coliseum – a period filled with fear and ignorance. These systems were designed to control people and thus prevent true accountability and ironically undermine effective control.

Many managerial systems are based on now discredited assumptions about human behaviour; assumptions discarded forty years ago in the 1960s or even earlier.
Derailed leadership
Partly because of these counterproductive systems, traditional styles of leadership are out of touch with the diverse and dynamic reality of organisational life. Traditional styles of leadership are often based on a lack of understanding of core processes and human behaviour. Conventional approaches to improving productivity and managing change often have limited impact or, worse, can be ineffective and even detrimental. People want and deserve much more.

Sadly, although many humans naturally yearn to continually improve tasks and processes, in some organisations increasing productivity has become something that stimulates fear. The stress and resulting physical and emotional damage is widely recognised as a huge cost with ongoing and long lasting consequences. This is a terrible waste of resources at a time when high productivity is vital.

UNDERSTANDING HUMAN BEHAVIOUR
There is an easier and more effective way to manage organisational change successfully. This is based on first changing behaviours, which is relatively simple and straightforward. Consequently, at an organisational level, instead of living in the past, leaders need to build and use systems that live in the present so they are conscious of how they use their own energy and other people’s energy.

The main drivers of human behaviour and underlying needs
In a practical sense there are five main drivers of behaviour. These are:
1. genetic makeup,
2. personal patterns developed in early years of childhood,
3. past work experience,
4. the organisation’s systems, and
5. leadership.

These five broad drivers of behaviour determine how a person’s energy is used and, importantly, influence whether a person’s response to a situation is made from conscious choice or unconscious, habitual reactions. Whether the person’s energy is used consciously or unconsciously, all behaviour is motivated by people’s attempts to fulfil an underlying unmet conscious or unconscious need or maintain fulfilment of needs currently being met.

Turning now to the first driver of behaviour – genetics. Clearly once a person is hired it is too late to do much about genetic makeup! For this reason, the paper will move now to discuss the remaining four broad drivers of behaviour.

Habitual unconscious personal reaction patterns
Over a period of half a century Maria Montessori (1948) made the most extensive and detailed observations of human development during childhood. Her discoveries included her observation that ‘the principal years for formation of both character and intellect are from zero to six’. This early formative period’s primacy in developing reactive patterns is confirmed by science’s latest discoveries in brain development and human behaviour (Goleman, 1996; Childre, 2000). These and other scientific advances confirm the teachings of Buddha 2500 years ago and observations by the ancient Greeks 3000 years ago.

Reaction patterns are built into the formation of the brain during childhood. A person’s reactions to situations at work, in the home and in the community are largely built before puberty and especially during the first six years.

The human brain’s rational neural network is designed to search out patterns in the person’s environment. This inherent ability is used in maths, language and behaviour. Children pick up behaviour patterns from their environment, especially from significant adults in their lives. The adult neural network carries patterns of behaviour shaped over many years, especially during early childhood.

As well as picking up patterns, in every moment of their life people have feelings coursing through their body. These feelings are intimately associated with the release of internal chemicals such as hormones, internal electric currents and electromagnetic fields, variations in pressure within the body and other means of internal bodily communication. Feelings trigger these movements of energy within. Feelings are a core part of each person’s internal communication system.

Clearly, humans are very complex creatures – yet the basics are simple. The brain consists of neurones which extend throughout the body as the nervous system. Nerves are enmeshed in tissues – intimately part of almost every fibre of the human being.

Humans are hard-wired with a fight-or-flight reactive mechanism. It can cause even highly intelligent, highly rational people to be hijacked by their emotions; for example: road rage, screaming at the kids, the boss going off his brain, a partner sulking or withdrawing. These reactions are often unconsciously triggered by underlying anxiety or fear. As a result, energy is wasted unconsciously for no productive or personal benefit.

Individual learned coping mechanisms seen as habitual reactions to a situation could include shutting down, shying away, embracing changes, rejecting changes, laughter, aggression, denial or submission to name just a few. In response to an event or situation two people can experience contrasting feelings. One may feel happy, the other upset. One may feel stimulated, the other overwhelmed. One may become aggressive, the other run away and yet another may be calm and cool. Clearly happiness, sadness and indeed all feelings come from within. Feelings are generated within.

Essentially for many people, reactions are not conscious, ie their energy is directed unconsciously. Yet as Goleman (1994) and Thomsen-Moore (2004) state, emotional intelligence is a far more significant factor in personal performance than is IQ. Emotional intelligence can be summarised as the ability to be self-disciplined, develop understanding and empathy for oneself and others and develop personal connection with others. Essentially, in the workplace emotional intelligence is about using feelings intelligently to detect, understand, manage and express feelings and emotions productively.

Underlying feelings are associated with met or unmet needs. In essence, on a personal level, needs give rise to feelings which in turn drive either unconscious emotional reaction or, in some more aware individuals, conscious choices.

Note that use of the word feelings is not meant to be associated with Hollywood’s use of the word as mushy, soft, vague underpinnings of vulnerable emotions. In reality, at every moment of their lives all humans experience feelings which reflect real movement of energy within the body. Their conscious or unconscious response to such feelings determines personal effectiveness.

When feelings are not recognised by a person’s conscious awareness, that person can experience reactive e-motions being ’energy in motion’. These emotional reactions are patterns developed in early childhood and can represent substantial waste or blockage of energy. Alternatively, when people are conscious of underlying feelings coursing through their body, they can then link them to underlying real needs and thereby consciously make choices on how to use their energy most productively.
In this sense, consciousness or conscious awareness is the root of ‘response-ability’, which simply means the ability to choose a response. Additionally, feelings affect natural vibration frequencies within the body’s primary electrical generators – the heart and brain – which continually radiate electromagnetic energy. Such energy can be measured up to three metres from the heart. It can be transmitted large distances via changes in vocal tone and via micro-muscular and colour changes in the face. Childre and Martin (2000) discuss how the frequency of such vibrations depends on the feeling experienced. For example, anger generates a vastly different vibration frequency than do feelings of calmness and confidence.

Humans automatically broadcast or radiate these feelings. Humans automatically receive or take in these feelings from others. In essence, the human body is a natural antenna for broadcasting and receiving energy in the form of vibrations associated with specific feelings. In this way, feelings within one human can radiate and trigger feelings and unconscious reactions in the human receiver.

Many people experience a loss of personal power in organisations, especially when there is a lack of consciousness of the source or true cause of their feelings and reactions. By identifying and acknowledging feelings though people can reclaim their power. This is why leaders need to understand unconscious human behaviour.

**Past work experience**

Clearly people bring to a current employer their experience developed in previous work environments. This influences their behaviour, or use of energy and time. For now, there is no need to expand on this other than to say that in the absence of clearly defined systems a new employee will tend to maintain behaviours developed in the past in previous organisations.

**Systems drive behaviour which shapes attitude**

As has already been established in this paper, systems drive behaviour. Remember the earlier example – if, as managers, we want to team-work, yet measure and reward only individual performance, people will focus on individual performance.

A significant example of systems driving behaviour and changing attitudes is the impact of random breath testing to curb driving under the influence of alcohol on Australian roads. Prior to the introduction of random breath testing the attitude in Australia toward drink-driving was generally one of indifference bordering on pride. To reduce drink-driving, state governments tried logical advertising quoting, for example, fatality rates associated with drink-driving. That failed dismally. Next they tried emotional advertising since as many advertising executives, union delegates and politicians know emotional messages have much stronger impact on people. Emotional messages raised awareness but did not change drivers’ behaviour.

Victoria then became the first state in the world to introduce random breath testing of drivers. Immediately, behaviours changed and noted Australian social researchers such as Hugh Mackay now attribute a substantial change in social behaviours and attitudes to the introduction of random breath testing.

It is important to emphasise that the legislation did not attempt to change attitudes. It merely put in place a system that changed behaviours. Subsequently, attitudes changed to align with behaviour. Humans do not go through life with a certain belief while behaving contrary to that belief. Rather, if the changed behaviour is maintained, experience and scientific research confirm that attitudes change to align with and justify behaviour. This effect has been used by many organisations including political parties and unions to develop behaviours and attitudes aligned with the needs of the organisation and/or its leaders.

Please note that although the example of random breath testing is an example of a punitive system, the use of process-based systems drives productive behaviours constructively.

The conclusion is firstly that communicating and/or training alone are not adequate. Secondly and importantly, system change does drive behaviour change. Thirdly, when system change is supported with effective training and communication such behaviour change is accelerated.

Deep down, intuitively, many managers know systems drive behaviour. Yet many just don’t seem to know it on-the-job. Instead, many managers seem to be trying to buy change off-the-shelf as a one-size-fits-all package without looking within to understand. That’s unconscious.

To change organisational culture it is necessary to change the systems driving behaviour. While it is difficult, indeed impossible to change people, it is possible to change behaviours which will lead to a change in attitude. A change in behaviour and attitude produces a change in culture.

**Leadership**

Leaders through their own modelling of desired behaviours provide a positive example. People tend to emulate the behaviour of their leader or at least focus on what their leaders focus. This applies especially with leaders people respect and trust. On the other hand, if leaders ignore people and focus mainly on, for example, technological aspects, most people throughout the organisation perceive a lack of importance of people issues. This has important consequences in management of safety and productivity.

For instance, it is becoming increasingly clear that leaders focusing blindly on cost-cutting can consciously or unconsciously drive people to cut costs in ways that are not cost-effective, ie in ways that reduce productivity.

**PRACTICAL SOLUTIONS**

In summary then, to effect culture change and to successfully manage organisational change, leaders need to be aware every individual has deeply ingrained personal patterns that can be very difficult to shift. With one exception discussed below, it is not possible to change people. Rather, individuals change themselves in response to personal searches for consciousness. In groups though, people change behaviour in response to changes in systems accelerated with effective communication and modelling by leaders. This behavioural change then leads to attitude change and thus culture change.

Leaders particularly need to become aware of their own learned personal patterns which shape their leadership style and influence the systems they design. In this way, the leader’s awareness of personal patterns has many impacts on the whole organisation.

Instead of the outdated, flawed and ineffective HR model, approach behaviour from the perspective of energy consciousness and energy use, ie use a practical approach of changing systems to change observable behaviour.

**Improving organisational behaviours and attitudes**

To improve productivity and safety, leaders need to observe people’s behaviours to understand use of energy in the organisation. Then identify systems driving behaviours. These are occasionally designed consciously. Many more are often built deliberately and yet unconsciously. Others just grow in an ad hoc way in response to various past executive decisions, past events, unions and other influences. To be effective, leaders need to define specific desired behaviours and then design, build and implement systems to drive those desired behaviours.
By way of emphasis, it is necessary to restate that by far the most powerful system for driving behaviour is the measurement, analysis and reporting system. Because it is associated with people’s sense of achievement and worth it is even more powerful than remuneration systems, including performance bonuses and monetary incentives.

Other basic systems include organisation structure (including roles and responsibilities), communication systems, planning systems, behavioural and process standards, personal feedback and performance development systems. Systems on the next level of drivers include remuneration systems, systems for involving people, recognition systems, methodologies for process and productivity improvement and the organisation’s plan for organisational change. The detailed interaction and integration of these systems is beyond the scope of this paper and is introduced in Roberts (1995). Additional systems include, for example, safety systems, administration systems, selection and preparation of people (recruiting), office layouts, computer systems, environmental management systems, quality assurance systems, document systems and policies and procedures.

To drive productive behaviours aligned with the business’ core processes and purpose leaders need to consciously design and build simple, solid conscious systems to drive behaviours aligned with and supporting core processes and the business’ purpose.

To describe these systems in detail here is beyond the scope of this paper. Examples of their benefits though include the more than doubling of roadway drivage rates in an underground nickel mine through adoption of process-based measurement, analysis and reporting. At an American dredge mining operation an increase in production equipment availability from 89 per cent to 94 per cent through changing the organisation structure immediately produced over two million US dollars in annual benefits from increased recoveries and additional unmeasured benefits in terms of greater teamwork at all levels.

Take the example of aligning measurement, analysis and reporting with the core process and aligning all basic systems to focus on performance improvement and teamwork. In the early 1990s Australia’s largest longwall mine achieved development rates close to double the next best Australian performance under similar conditions. It also achieved longwall face productivity 30 per cent higher than the next best Australian face. Significantly, this represented productivity double that of the next best mine operating under similar conditions. These records were achieved with a performance bonus system. Industry figures in NSW and Queensland show the mine enjoyed by far the best safety performance of all large underground coal mines in Australia. This was driven essentially by leadership commitment to safety and the use of proactive safety performance measures.

A small longwall mine doubled productivity with reduction in equipment used. A large Australian surface mine increased recoveries three per cent, it while simultaneously improving throughput and industrial harmony. Such improvements go straight to the bottom line.

To successfully manage organisational change leaders need to develop a plan for changing systems. After all, most people, including managers would not build a building without a plan. Yet even though culture has a far greater impact on productivity than do buildings, the use of project management techniques and process change systems to build productive cultures is rare. Such use though is proven in practice to be highly effective.

Additionally, solid, well considered plans reduce people’s uncertainty and fear since such plans meet people’s needs for certainty by providing direction, clarity, reassurance and confidence.

Measurement, analysis and reporting of performance

Conventional measurement, analysis and reporting systems drive reactive behaviour and/or apathy and stimulate counterproductive corrosive or benign leadership. Even though accurate understanding of variation is the key to accurately identifying the greatest opportunities for productivity improvement, conventional measurement, analysis and reporting ignores variation and misleads managers and executives.

Instead, to be effective the measurement, analysis and reporting system needs to provide an understanding of variation and clearly identify the two main causes of variation – inherent natural variation and process change. This is the key to effective deployment of assets, especially executive time and energy. Such a system provides better understanding of performance. It enables quicker and more accurate identification of opportunities for improving productivity and dramatically increases executive productivity. Significantly, it improves managerial behaviours so that managers focus more on leading and supporting processes and people within those processes. At all levels within an organisation it develops true accountability.

Developing leadership and personal emotional mastery

When building systems it is important to recognise the impact of personal energy in at least three ways. Firstly, whether or not it is intentional and conscious, a good building systems, a leader’s personal patterns will affect the systems built.

Secondly, use of a solid project plan for managing organisational change will develop confidence in the executive team. This confidence radiates from the executives and naturally transmits to people throughout the organisation.

Thirdly, acknowledge the importance of interpersonal connection by building it into the formal and informal communication systems. While noticeboards are read on average by only around five per cent of people and written personal correspondence is more effective the use of personal oral communication has much more impact. Much more powerful again is face-to-face personal communication.

Of greater power and effectiveness are leaders’ actions, energy and feelings since these are broadcast throughout the organisation.

Leadership implies the provision of direction. For this leaders need to have the self-discipline to pause and consider how to balance strategies to meet the organisation’s needs for development. Leaders then need to develop plans for organisational and performance improvement. As this involves intangibles it can be difficult for some managers and executives to overcome reaction patterns and develop sufficient discipline to build and communicate sound plans.

Effective leadership requires consciousness to develop the discipline to take charge constructively and, where necessary, collaboratively to shape the organisation’s future. Often this requires tenacity, resilience and commitment to persist with the plan, particularly when facing obstacles. It requires enthusiasm and passion for the plan and for the organisation since personal energy connects and moves others.

Importantly, a leader’s personal patterns display the leader’s priorities to all members of the organisation. Thus, to be more effective leaders need to develop consciousness by exploring their personal behaviour patterns by identifying, acknowledging and managing underlying feelings, needs and deeply held values.

Gendlin (1981) provides understanding of the presence within humans of a felt sense. Indeed, Einstein credited his success to his ability to go beyond his rational intellect by using his complete intelligence. It is becoming increasingly important for leaders to explore the innermost recesses of their being to identify, explore and manage personal patterns and intelligence. The technology for this is now readily accessible and many methods are available to explore the connection of feelings, intellect, physical and spiritual dimensions. Science is now
recognising the power of methods employing all these dimensions or modalities to uncover and change habitual counterproductive personal behaviour patterns. Ironically, modern science is now verifying the power of ancient techniques used for millennia to identify, explore and manage personal patterns. Increasingly, science, western medicine and ancient eastern philosophies and practices are aligning on shared paths. Dr Deepak Chopra (1993), a western trained doctor with strong roots in practices of ancient India refers to:

...a number of scientific studies that show the beneficial effects of meditating. Blood pressure comes down. Stress is alleviated. Basal metabolic rate goes down. Insomnia, anxiety, and a number of psychomatic disorders are relieved and disappear. Moreover, there is increased brain wave coherence, which also improves attention span, creativity, learning ability and memory retrieval.

These benefits combined with vastly improved immune system health are listed in research by the Institute of HeartMath whose work is now used to improve the effectiveness of soldiers in all four branches of the USA armed forces and the effectiveness of managers in many international corporations. It has been used in acclaimed research hospitals to replace conventional medical treatment.

Ironically, the 2500 year old technique of Vipassana meditation is now being verified by modern science for the technique’s success in identifying and changing deep-seated personal behaviour patterns. This is arguably one of the most powerful methods for improving personal effectiveness, relationships and personal productivity since it develops mastery over reactions by bringing patterns to the surface consciousness. Studies indicate it appears to dismantle ingrained patterns developed in childhood and builds new more effective patterns to give people choices when faced with difficult situations.

Psychiatrist, Dr Marshall Rosenberg (2000), has developed a process now referred to as Nonviolent Communication currently achieving success and recognition around the world for its incorporation of feelings and needs in a very natural way that improves communication effectiveness. This is particularly so in challenging situations such as addressing discipline when many managers explode with anger and aggression or retreat and withdraw in fear. There is a third approach in which such situations can be changed to be positive for both the deliverer and recipient of disciplinary proceedings. Like Gendlin, Rosenberg discovered innate abilities in humans for enhanced connection and overcoming patterns of separation that prevent effective communication within and between individuals.

Practical, objective and comprehensive personal feedback tools based on observable behaviours can be of immense benefit in assisting managers, executives and directors to identify and better understand personal habitual patterns. Such data is of immense benefit in improving personal and organisational productivity and happiness.

Emotional mastery and consciousness is no longer considered a soft option. It’s now about concrete, practical learnable skills and about being effective. Effective leaders are people who know themselves and consistently demonstrate self-discipline, emotional mastery, drive and connection with others. Fortunately, unlike intellectual intelligence, emotional intelligence can be enhanced, learned and developed.

This has significant impact on educators and legislators. Leblanc (2004) provides highly respected international research into executives and directors in many nations. His acclaimed work is exposing traditional corporate governance legislative approaches as not only inadequate, but undermining effective corporate governance and accountability. New Canadian corporate governance legislation instead considers the actual drivers of behaviour by executives and directors to increase accountability.

Sound industrial relations rooted in a solid managerial framework

In many underground and surface mines, particularly within the coal sector, industrial relations can provide challenges. In such cases experience shows it is necessary to have a firm approach to industrial relations supported by a solid corporate philosophy and solid corporate leadership. The importance of a clear vision, direction and constancy of purpose remain, as always, vital.

Combined with the personal energy aspects covered above, industrial relations can be turned to a strength for making and sustaining improvements while simultaneously building an organisation’s capability.

Proven methodology for improving safety and productivity

Many organisations blindly embark on process and productivity improvement campaigns that are counterproductive or sustainable only with constant managerial attention. Instead, when leaders use the methodology of improving productivity by systematically reducing variation to control and stabilise processes, future improvements are locked in. Importantly, it becomes easier to continually improve both by making incremental improvements and by substantial step changes.

Allied with this is the need for measurement, analysis and reporting systems that enable rapid and reliable understanding of variation. Such methods rely on graphical presentation of data using simple statistically sound methods of analysis. These methods are usable by people of all levels of education. More importantly, these methods drive productive, supportive behaviours in leaders while enabling them to develop much greater accountability throughout the organisation.

Building the organisation’s capability

Enduringly effective leaders focus on much more than simply improving today’s productivity. They simultaneously build solid organisations in which continual productivity improvement is a normal part of business. Collins and Porras (2004) provide studies supporting use of aligned systems to ensure an organisation’s future productivity.

TAKING RESPONSIBILITY FOR SAFETY AND PRODUCTIVITY

To be effective, leaders need to take responsibility for the use of their energy and the energy of the people they lead. At an organisational level, effective leaders understand the drivers of organisational and group behaviour. They take action to understand and build conscious, process-based systems to drive desired organisational behaviours and shape attitudes.

At a personal level, effective leaders develop conscious awareness of their personal patterns. To achieve this, executives, directors and managers explore their doorway to emotional mastery for inner peace and higher sustainable productivity. That doorway is often consciousness of deep feelings and needs. Opening the door by exploring underlying feelings, these deep feelings then become the path and behaviours become signposts. This is the road to consciousness – the foundation of accountability for superior productivity and safety.

Effective leadership involves the conscious, committed use of human energy. At all times leaders need to be conscious of the drivers of their personal behaviour and the behaviour of groups and organisations they lead.
The choice is not whether leaders address their responsibilities. An increasingly competitive global market and demanding stakeholders ensure the only choice is when leaders face up to organisational and personal consciousness.

REFERENCES
Health Tracking Project — The Development of a National Framework for Managing Occupational Illness and Disease in the Australian Minerals Industry

C Bofinger

ABSTRACT

The Health Tracking project is one of five projects being undertaken as part of the Minerals Industry Cooperation Initiative (MICI) – a national initiative sponsored by the Minerals Council of Australia. The aim of the Health Tracking project is to assess the practicability of ways to demonstrate the monitoring of hazardous exposures and the occurrence of related occupational illness and disease and development of management systems and strategies at a national level.

This second stage of the project will be completed in 2005 and will cover four areas:

1. pilot development of a comprehensive job exposure matrix (JEM) for the minerals industry;
2. development of both proactive and reactive occupational illness and disease metrics;
3. recommendations regarding appropriate health tracking models; and
4. provision of best practices guidelines for management of occupational health.

The synergies between these areas will establish the path towards an effective national occupational health surveillance system and the development of effective prevention strategies and policies.

INTRODUCTION

The Health Tracking project is one of the projects under the Minerals Industry Cooperation Initiative – MICI (Bofinger, 2004). MICI projects include the Health Project, Lessons Learned, Professional Pathways, MIRMgate and National Minerals Industry Risk Assessment Guidelines. The intent behind the projects is to address factors impacting on the occupational health and safety risks in the industry and to demonstrate, by 2006, that cooperation and the sharing of resources and information between mining companies is achievable and valued. This will form the basis of discussions with other industry stakeholders for a broader cooperative initiative.

The National Occupational Health and Safety Commission (2000) reported on broad issues of occupational health and safety data in Australia and concluded that the overall health burden of occupational disease was much greater than that caused by injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury. Occupational disease was grossly under-reported because the current data systems were ineffective in capturing data on injury.

There is no comprehensive system of surveillance for occupational disease and illnesses. Surveillance is vital to the prevention of occupational diseases, injuries and fatalities. It provides information necessary to draw attention to the magnitude of workplace health and safety problems, to set research priorities and to target and evaluate interventions to improve worker safety and health. The current situation is fragmented with information relevant to the minerals industry held by:

- companies and sites,
- medical personnel, and
- workers compensation.

Information to allow comparison of the health status of miners with other industries and the general population is held by the Australian Institute of Health and Welfare.

Analysis of the national compensation dataset compiled by the National Occupational Health and Safety Commission (NOHSC, 2002) shows hearing loss and musculoskeletal disease are the most common reported occupational illnesses. These are also relatively easy to identify and well known problems in the mining industry. However, information on other health issues is very limited.

Figure 1 shows the frequency rate for illness or disease for 2000 - 2001 for mining related industries (NOHSC, 2002). The frequency rate is the number of occurrences expressed as a rate per million hour worked by wage and salary earners. The total number of diseases or illnesses reported for mining is 545. The number of claims and the frequency rates reported for occupational illness and disease are generally low and care should be taken in interpretation of results.

The frequency rate for diseases of the nervous system and sense organs is highest for the coal mining industry. Diseases of the nervous system and sense organs are the highest frequency rate for the metalliferous industry but it is 0.7 compared to 4.5 for coal mining. The frequency rate for all industry is 0.6.

An examination of the published literature available revealed some analysis of occupational health or occupational disease issues for the Australian mining industry. There are limitations associated with the types of analysis reported in the published literature. The data is generally averaged, summarised, and de-identified. This type of analysis does allow trending of some diseases and illnesses for the mining industry or some sectors of the industry.

Coal industry

- Vibrating and jarring causing injury in the New South Wales coal industry (Cross and Walters, 1994) – This work examined the compensation data and found no evidence that whole body vibration was a significant factor in head, neck and back injuries.

- Emphysema and lung content (Leigh et al, 1994) – The results of this study showed strong evidence that emphysema in coal workers was causally related to lung coal content and therefore exposure to coal in life. The importance of age and smoking in severity of emphysema was also confirmed. A similar study was completed for chronic bronchitis (Leigh et al, 1986).

- Mortality in the New South Wales coal industry (Christie et al, 1995) – Generally the mortality was lower than the general population and a ‘healthy worker’ effect may explain the lower overall mortality.

- Cancer in a New South Wales coal miners (Brown et al, 1997) – This study considered the incidence of cancer in coal miners between 1973 and 1992 and concluded there did not appear to be a general risk of cancer in the NSW coal industry but that open cut miners have an increased risk of malignant melanoma.
• National Workplace Health Project and Queensland Health at Work Project – The national project compared lifestyle information affecting health between coal miners and other industry groups (Harris et al., 2000). The Queensland project gathered and analysed information from Queensland coal mines (Parker et al., 1996).

• Heart disease and coal mining (Bofinger and Ham, 2002) – There does not appear to be an increased risk of death from heart disease for the Australian coal industry. There are a number of factors that indicate an increased risk of heart disease for coal miners. This study included some industry summaries of occupational and lifestyle factors affecting heart disease.

Metalliferous and extractive industries

• Mortality in gold and coal miners and emphysema in Western Australia (Armstrong, 1979) – Overall, neither gold nor coal miners have a significantly higher mortality than expected from the experience of WA males in general. Lung cancer mortality was higher than expected in gold miners. Coal miners showed a lower than expected rate of lung cancer but an excess of deaths from other forms of cancer.

• Respiratory disease in goldminers in Western Australia (Musk, 1992) – Respiratory abnormalities were identified in non-smoking underground gold miners. These results were related to duration of employment, after adjusting for age and height, and were consistent with the presence of airway narrowing or emphysema.

• Asbestos and silica related diseases in Western Australian gold miners (Lee et al., 1999; de Klerk and Musk, 1998) – Asbestos-related pleural disease has been diagnosed in a small number of gold mine workers with no other significant known asbestos exposure. Gold miners were monitored to examine the relation between respiratory symptoms, smoking habits and employment history and the development of silicosis and lung cancer.

• Respiratory disease in bauxite miners (Beach et al., 2001) – This study determined that there was little evidence of a serious adverse effect on respiratory health associated with exposure to bauxite in open cut mines in present conditions.

• Mesothelioma in different occupational groups (Yeung et al., 1999) – Although Australia has one of the highest national incidences of mesothelioma in the world, the traditional primary asbestos industry cases from crocidolite mining and milling are now on the decline.

• Thorium and mineral sand workers (Hewson and Fardy, 1993) – This study was designed to complement estimates of radiation dose derived from air sampling measurements. It concluded that such doses must be interpreted with caution. The image of the mining and minerals industry as hazardous to health persists in the community. In order to meet external community expectations of management of health issues, the industry needs to demonstrate proactive involvement in this management. This was the main driver for this project.

INITIAL PHASE OF PROJECT

The initial focus of the project was to identify and analyse the different approaches to occupational health monitoring and surveillance currently in place and identify the influence of the current situation on the potential for a national system (Bofinger, 2004).

Phase one

Mindful of the definition of health surveillance used within the project as being:

\[
\text{Health surveillance is the ongoing systematic collection, analysis and interpretation of data for purposes of improving health and safety.}
\]

we identified the limitations of the existing situation.

Recognition of need for health surveillance

The need for some form of health monitoring of individuals is well recognised and widely practised throughout the mining and minerals industry.

Recognition of the need and requirements for health surveillance is less well recognised but is growing.

Identification of outcomes required

The Government schemes in place were established as a result of the history of disease in the mining and minerals industry. The focus for these centralised schemes is occupational health information. There are differences in the philosophies behind the schemes and the data collected.

Current medicals completed under Government or company schemes are unlikely to identify the physiological changes that occur at the early stages of an occupational disease affecting other systems due to latency of onset and limited diagnostic criteria. There are exceptions, eg where biological monitoring is conducted in the lead industry.
The focus of company schemes was generally to prevent or rehabilitate injury, or to ensure fitness for duty.

Outcomes for both the company and government schemes, in terms of identifying occupational disease and illness, have not been clearly defined.

**Identification of minimum data set**

As the outcomes of the schemes have not been clearly defined, the type of data collected is often unsuitable or incomplete.

There is inconsistency in the identifying data used and this limits the cross-linking to other data sets. Privacy legislation impacts on the data collected.

There is limited exposure data held electronically and little or no correlation between health information and exposure data either at the Government or company level. The data currently held in electronic data sets limits both the following of individuals and the identification of trends.

**Data capture mechanisms**

Electronic data capture mechanisms are limited for company schemes. Government schemes hold electronic records.

There is a need for company schemes to be organised so that data can be analysed to establish trends and allow following of individuals. This includes the electronic storage of health and exposure data.

The limited exposure data that is collected in an electronic data set makes it difficult to establish a relationship between occupational exposure and disease particularly when there may be lifestyle factors that also affect the likelihood of disease.

If electronic data capture is to be more widely established, consideration needs to be given to privacy concerns, costs and resources and the potential for litigation for compensation.

Compatible data capture mechanisms using a consistent data set need to be developed.

**Analysis and reporting requirements**

There is limited analysis and reporting from the current schemes. This limits any ability by companies or others to manage the risks. Information currently collected from medicals and workers compensation, if available in a suitable form, could be used pro-actively to improve the health of workers in the mining industry.

Occupational health surveillance data in the Australian mining industry is fragmented, collected for different purposes and limited analysis has been undertaken to enable the industry to identify or respond to illness or disease. However, company systems and legislative schemes can provide a vital foundation for a national system.

**Phase two**

In phase two of the project, four areas have been identified as being appropriate to progress as part of the MICI health project. At the end of phase one, advice was sought from the Health Working Group of the Minerals Council of Australia on the direction forward to ensure appropriate and useable outcomes. Work in the following four areas is currently in progress.

1. scope out SWOT for tracking models that are available;
2. investigate the potential for developing a comprehensive job exposure matrix (JEM) for the minerals industry;
3. development of occupational disease and illness metrics; and
4. provision of best practices guidelines for management of occupational health.

**Scope out SWOT for tracking models that are available**

This outlines the models that are available within Australia and internationally and identify the strengths, weaknesses, opportunities and threats (SWOT) of the current systems. It will provide information for decision making on the type of potential models.

The two Australian models that have been considered are Health Watch (for the petroleum industry) and Health Wise (for the aluminium industry). Detailed discussions have been held with the organisations managing the systems and the end-users about the benefits and limitations of the systems (Bofinger, 2004; Health Watch, 2000).

For phase two, the suitability of a tracking system for the occupational illnesses and disease that have been identified as being a priority for the industry based on industry input and consistent with the NOHSC priorities will be considered. These will include:

- noise induced hearing loss,
- respiratory illness,
- musculoskeletal, and
- cancers.

**Investigate the potential for developing a comprehensive job exposure matrix (JEM) for the minerals industry**

A JEM for the minerals industry (MINEJEM) was considered to be potentially beneficial for the industry. This is under further investigation and is initially focussing on the following areas:

- noise; and
- hazardous substance exposure leading to respiratory illness or disease:
  - respirable dust,
  - dust,
  - silica, and
  - heavy metal, eg lead.

The data collected as part of the state-based exposure systems will be considered. These areas and the data available were suggested as a ‘pilot’ study to determine the practically of the system and to allow working out of problems with the system. Information is available on the exposures of these hazards. The challenge is gathering the information and assessing the quality and value of the monitoring information.

JEMs cross-tabulate classified exposure information by chemical agent and occupational class. JEMs are also applied as a general exposure information systems for hazard control, risk quantification and hazard surveillance. The system includes, eg workforce data, and it provides information on the numbers of exposed workers by agent, occupation, and level of exposure. Exposure is described by the prevalence of exposure, eg per cent of workers exposed and the level of exposure among the exposed.

The detailed information that will be considered relating to the data to be used in MINEJEM will include:

- definition of jobs and tasks and what classification is used,
- type of data available,
- quality of data and how this would be assessed,
- methods used for gathering data,
- front-end format for MINEJEM, and
- Occupational Exposure Limits (OEL) that would be used.
The European Chemical Industries Council (CEFIC) has been developing an exposure database for the past four years (Cherrie and Kromhout, 2004). This is being provided to the Australian Minerals industry for evaluation in the first half of 2005. This database could form the base for the JEM.

**Development of occupational disease and illness metrics**

The aim of metrics or indicators is to provide different stakeholders with information on the effectiveness and efficiency of the management of occupational health and occupational illness and disease.

Occupational health has the potential to be defined by lead indicators. The applicability and appropriateness of lead indicators for occupational illness/disease is open for debate.

There are some general principles governing the choice of indicators that need to be considered:

- indicators should include more than one data source including integration of additional data sources in addition to mainstream outcome health factors, eg:
  - risk factors, and
  - behaviours.
- the focus should be on measuring change as the estimates of absolute levels will vary as information sources evolve and more detailed information becomes available; and
- regular reporting should be undertaken – similar to the current situation for safety statistics.

It is anticipated that the metrics will be based on three areas covering:

- OH policy and infrastructure,
- working conditions, and
- health outcomes.

This would lead to the development of metrics suitable for the Australian minerals industry. These metrics need to include both proactive and reactive measures and will focus on the acute and semi-acute measures – not the long-term measures. It would allow proactive tracking of the management of illness and disease.

**Provision of best practices guidelines for management of occupational health**

The fourth part of phase two involves interaction with another of the MICI projects. MIRMgate is a metadata system designed to provide high quality information on risk management in a searchable format for the minerals industry.

Information relating to health monitoring and health surveillance is present on the system and as new sources of information become available, these are added. Full details of the MIRMgate system are available on the MISHC website (MISHC, 2005).

A model for demonstrating the interaction of the different stages of the health project has been developed and is shown as Figure 2. Work is being completed in the different areas. This will be used to develop potential metrics to demonstrate management of occupational health from both proactive and reactive aspects.

**SUMMARY**

The synergies between these four areas will establish the path towards an effective national occupational health surveillance system and the development of effective prevention strategies and policies. The minerals industry will be able to proactively demonstrate management of both occupational health and occupational illness and disease.

The health project as part of MICI is a work in progress. The success of the project is dependent on the cooperation of the different sectors of the industry.

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**Fig 2 - Model for health tracking project.**
REFERENCES


Management Systems for Hazardous Exposures — Evidence of Failure and Opportunities for Success

B Ham¹

ABSTRACT

Coal mining legislation makes frequent reference to ‘Safety Management Systems’ but there is a chronic shortage of useful guidance material as to how these might be applied to hazardous exposures. Several studies have indicated that for each death that is reported as a result of an industrial ‘accident’, there are five deaths that caused by occupational exposures. Because the deaths from occupational exposures do not occur on site and often well after employment has ceased, they are generally unreported. Improved access to mortality data from coroners, the Australian Institute of Health and Welfare and superannuation service providers (early claims from death and total permanent disability), has given rise to some analysis of death and disability data that provides evidence that there are emerging opportunities to develop and improve safety management systems for hazardous occupational exposures. Issues explored include how safety management systems might be developed in relation to dusts and chemicals and other hazardous exposures to ensure the risk of disorders with long latency periods are reliably assessed and the exposures effectively managed. The current standards and guidelines which refer to safety management systems in general or to management of exposure related hazards are examined. Many standards are based on dose-response studies that provide for an acceptable percentage of workers to suffer adverse health outcomes. An alternative management system might be to apply dose-response relations to health surveillance to identify ‘at risk’ individuals and effectively manage health and related safety risk at an early stage.

INTRODUCTION

The question needs to be asked – ‘What should we be doing now to protect our work force from the long-term effects of hazardous exposures that the Western Australian asbestos mining industry should have been doing in 1960 if they were working under our current legislation?’. In a simplistic way, the answer is not complex – we record exposures, we seek data on adverse health outcomes, we examine the dose-response relationship and implement controls which ensure the risk of an adverse health outcome is reduced to an acceptable level.

The reality is that the process is very complex, takes many years to resolve and extraordinarily difficult to implement. In terms of a threat to sustainability, the asbestos mining industry can provide a lesson well worth learning.

LEGISLATION

The Queensland Coal Mining Safety and Health Regulation 2001, requires mine operators to implement a ‘safety and health management system for personal fatigue, and other physical and psychological impairment and drugs’ in section 42. The regulation also establishes the Coal Mine Workers Health Scheme in sections 44 to 53. This scheme provides for medical practitioners called Nominated Medical Advisers to promote safe operations by assessing workers’ fitness to undertake duties without risk to themselves or others. The Scheme also monitors changes in the health of mine workers over time. The key section that relates to hazardous exposures is section 49 which requires:

> A coal mine’s safety and health management system must provide for periodic monitoring of the level of risk from hazards at the mine that are likely to create an unacceptable level of risk.

The New South Wales coal industry is subject to the Occupational Health and Safety Act 2000 which establishes broad duty of care on all mine operators. When new regulations are finalised, the industry will also be subject to the Coal Mining Health and Safety Act 2002. Sections 20 to 22 refer to the obligations of mine operators to prepare, implement and ensure compliance with health and safety management systems. More specifically Section 23 specifies the Contents of health and safety management system:

1. The purpose of a health and safety management system must be to provide the primary means by which an operator ensures the health, safety and welfare of employees and others at a coal operation and of people directly affected by a coal operation, including people who are not at the coal operation.

2. A health and safety management system for a coal operation must provide:
   a. the basis for the identification of hazards, and of the assessment of risks arising from those hazards, by the operator of the coal operation;
   b. for the development of controls for those risks; and
   c. for the reliable implementation of those controls.

REVIEW OF THE EVIDENCE

Kerr et al (1996) indicates that disease related deaths are grossly under-reported and that for every reported occupational related death, there may be five further occupational related deaths that are unreported.

The study by Bofinger and Ham (2002) includes 13,000 mortality records of previously registered coal miners in New South Wales and Queensland for 1980 to 2000. The results for 1996 to 2000 are shown in Table 1.

The study shows elevated rates of cancers in Queensland workers and elevated heart disease in NSW miners but the most dramatic difference in the injuries from external causes where NSW is three times higher that the general population. A flaw in the study design is that the age profiles of the miners do not necessarily reflect the age profile of the general population and that the study did not extend to examining age specific death rates.

As a step towards examining death rates, the birth cohort from the death data of New South Wales miners is developed as shown in Table 2. This shows that from a register of 67,785 miners, matches could be found for 12,533 miners (1900 to 2001). The limited percentage of fatalities in the 50 to 80 age groups over the period 1920 to 1950 indicates a significant level of missing data due to poor matching or migration.

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The birth cohort table is used to estimate a survivor population for the calculation of death rates that can be compared with published AIHW data. Rates are calculated as deaths per 100,000 of population and are undertaken by cause, year and age group. The process is used to examine specific disorders such as lung cancer as shown in Table 3 and Figure 1. The lung cancer data indicates that lung cancer is rising in the coal miners while it is declining in the general population. The difference might be attributed to less effective uptake of anti-smoking health promotion in the mining population. There is some opportunity to reduce these errors by applying standardised population techniques.

Analysis of data from the Queensland Coal and Oil Shale Superannuation Fund (QCOS) by Ham (2003) explored early superannuation claims which provided an alternative source of death data. Table 3 shows 51 deaths and 216 cases of total permanent disability. Key issues to rise out of the analysis of the QCOS data are:

1. coding of mortality data grossly under-estimates the contribution of nervous and mental disorders to fatalities; and
2. fatality data does not account for the wide spread of total permanent disability suffered by mine workers.

An alternative approach to health performance indicators is to examine the median age of death by cause and group. This overcomes reliability issues with the estimation of population when calculating mortality rates. The data is shown in Table 4.

### Table 1

<table>
<thead>
<tr>
<th>ICD code No</th>
<th>Cause of death category</th>
<th>Number of deaths</th>
<th>% of deaths</th>
<th>Australian population %</th>
</tr>
</thead>
<tbody>
<tr>
<td>II</td>
<td>Neoplasms (cancer)</td>
<td>NSW miners 1996 - 2000</td>
<td>Qld miners 1996 - 2000</td>
<td>NSW miners %</td>
</tr>
<tr>
<td>IV</td>
<td>Endocrine, nutritional and metabolic diseases</td>
<td>70</td>
<td>7</td>
<td>3</td>
</tr>
<tr>
<td>V and IV</td>
<td>Mental disorders and diseases of the nervous system</td>
<td>64</td>
<td>6</td>
<td>3</td>
</tr>
<tr>
<td>IX</td>
<td>Diseases of the circulatory system</td>
<td>940</td>
<td>75</td>
<td>39</td>
</tr>
<tr>
<td>X</td>
<td>Diseases of the respiratory system</td>
<td>232</td>
<td>12</td>
<td>10</td>
</tr>
<tr>
<td>XI</td>
<td>Diseases of the digestive system</td>
<td>69</td>
<td>4</td>
<td>3</td>
</tr>
<tr>
<td>XIX and XX</td>
<td>Injury etc – external causes</td>
<td>149</td>
<td>66</td>
<td>6</td>
</tr>
<tr>
<td>All others</td>
<td></td>
<td>83</td>
<td>10</td>
<td>3</td>
</tr>
<tr>
<td>Total</td>
<td>All classes</td>
<td>2428</td>
<td>293</td>
<td>3</td>
</tr>
</tbody>
</table>

### Table 2

Birth cohort for New South Wales mortality data.

<table>
<thead>
<tr>
<th>Birth years</th>
<th>20 - 29</th>
<th>30 - 39</th>
<th>40 - 49</th>
<th>50 - 59</th>
<th>60 - 69</th>
<th>70 - 79</th>
<th>80 - 89</th>
<th>90+</th>
<th>Total deaths</th>
<th>Register</th>
<th>Deaths %</th>
</tr>
</thead>
<tbody>
<tr>
<td>1900 - 1910</td>
<td>Data missing – persons died prior to data collection</td>
<td>714</td>
<td>1042</td>
<td>267</td>
<td>2023</td>
<td>4553</td>
<td>44</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1910 - 1919</td>
<td>1</td>
<td>637</td>
<td>1819</td>
<td>867</td>
<td>10</td>
<td>3334</td>
<td>6759</td>
<td>49</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1920 - 1929</td>
<td>520</td>
<td>1817</td>
<td>1357</td>
<td>27</td>
<td>3721</td>
<td>11241</td>
<td>33</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1930 - 1939</td>
<td>234</td>
<td>881</td>
<td>915</td>
<td>29</td>
<td>2059</td>
<td>13101</td>
<td>16</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1940 - 1949</td>
<td>96</td>
<td>331</td>
<td>312</td>
<td>21</td>
<td>760</td>
<td>13195</td>
<td>6</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1950 - 1959</td>
<td>73</td>
<td>171</td>
<td>171</td>
<td>6</td>
<td>421</td>
<td>12968</td>
<td>3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1960/later</td>
<td>137</td>
<td>78</td>
<td>171</td>
<td>6</td>
<td>215</td>
<td>10521</td>
<td>2</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Subtotal</td>
<td>210</td>
<td>345</td>
<td>736</td>
<td>1720</td>
<td>3390</td>
<td>3919</td>
<td>1936</td>
<td>277</td>
<td>12533</td>
<td>67785</td>
<td>18</td>
</tr>
</tbody>
</table>

### Table 3


<table>
<thead>
<tr>
<th>Cause</th>
<th>Deaths</th>
<th>TPD</th>
<th>Totals</th>
<th>Av Age</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cancer</td>
<td>14</td>
<td>20</td>
<td>34</td>
<td>51</td>
</tr>
<tr>
<td>Circulatory disease</td>
<td>12</td>
<td>21</td>
<td>33</td>
<td>53</td>
</tr>
<tr>
<td>Ear disorders</td>
<td>0</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Endocrine disorders</td>
<td>0</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Infectious diseases</td>
<td>0</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>Musculo-skeletal disorders</td>
<td>0</td>
<td>83</td>
<td>83</td>
<td>47</td>
</tr>
<tr>
<td>Nervous/mental disorders</td>
<td>9</td>
<td>43</td>
<td>52</td>
<td>48</td>
</tr>
<tr>
<td>Respiratory disease</td>
<td>0</td>
<td>4</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td>External causes</td>
<td>13</td>
<td>32</td>
<td>45</td>
<td>41</td>
</tr>
<tr>
<td>Other</td>
<td>3</td>
<td>2</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td>51</td>
<td>216</td>
<td>267</td>
<td>48</td>
</tr>
</tbody>
</table>

**Fig 1** - Lung cancer mortality – 1995 to 2000 by year.
The younger ages in cancer and circulatory disease in Queensland miners may be partly explained by younger age distribution of the Queensland population. Follow-up work to correct for the age difference is warranted.

A key issue that arises out of the review of the health outcome evidence is the difficulty in measuring and quantifying adverse health outcomes. Some of the alternative approaches and their strengths and limitations are shown in Table 5.

### Standards on OHS Management Systems


While the Standard is lacking in detail, it provides a useful structure for the development of occupational health and safety management systems. The key components are:

- OHS policy,
- planning consultation, communication and reporting,
- documentation,
- document and data control, and
- measurement and evaluation.

The policy needs to be authorised and visibly supported by senior management and clearly state the OHS objectives and a commitment of improving OHS performance. The policy should:

- recognise the nature and scale of the organisation and its health risks;
- include a commitment to improving OHS and the OHS system;
- include a commitment to comply with relevant legislation and standards;
- be documented, implemented, maintained and communicated to all employees and contractors;
- be available to interested parties; and
- be reviewed periodically to ensure it remains relevant and appropriate to the organisation.

Planning needs to cover the identification of hazards and the assessment and control of risks. Planning also needs to take into account training, succession, contractors, legal and other requirements. In relation to health monitoring this includes compliance with workers compensation, privacy and anti-discrimination legislation.

While the Standard indicates that objectives and targets need to be established and implemented, some caution is required in relation to the limitation of many OHS performance indicators. Of particular concern is a focus on the lost time injury frequency rate. While a reduction of injuries is an admirable objective, there is a possibility that bonus programs based on this statistic may cause intentional under-reporting of injuries and incidents. Contractor performance monitoring puts them particularly at risk from this practice. The result causes an unidentified rise in the risk profile for the operation.

Reporting procedures should cover the following:

- OHS performance including results of reviews and audits;
- reporting of incidents and failures;
- reporting on hazard identifications; and
- reporting on preventative and corrective actions and statutory reporting requirements.

The organisation should establish, implement and maintain information to describe the elements of the management system and related documentation. The program including its documentation is then implemented and periodically audited and reviewed.
Western Australian Department of Mines and Petroleum Resources (MPR)


The objectives of the upgraded CONTAM system are:

• to provide comparative occupation group, industry sector, and industry exposure data and enable trend analysis of this data;
• to provide a reliable basis for future studies into the long-term health effects of exposure of mine workers to atmospheric contaminants; and
• to enable accurate assessment of company compliance in the maintenance of acceptable working environments.

To achieve these objectives, the new CONTAM system operates as follows:

Each mine will be required to submit a Workforce Survey Form to the MPR when requested. This form will provide the MPR with information on the number of employees, the type of work they do, and the contaminants they are exposed to.

The data reported on the Workforce Survey Forms will be used to calculate the minimum sampling requirements (quota) for each mine. Mines will be informed of their quota via Quota Allocation Reports which will be distributed by the MPR. Each mine manager and exploration operation manager will be responsible for ensuring the minimum sampling requirements are met. Sampling results will then be sent to the MPR on a CONTAM Sample Record Sheet, and entered into the CONTAM system. Sampling results will be used to prepare annual industry reports, which will be forwarded onto each mine.

Health surveillance program for mine employees – approved procedures, MPR (2002)

The objectives of the health surveillance program for mine employees are:

• to assess the health status of all mining industry employees on a regular basis;
• to analyse collected data to detect adverse health effects at the earliest opportunity;
• to enable appropriate and timely corrective action to be taken in order to safeguard the health and well-being of mining industry employees; and
• to provide data which may be useful for future epidemiological studies’.

The health assessments conducted for the Health Surveillance Program consist of work history; a respiratory questionnaire; a lung function test; an audiometric test; and in some cases, a chest x-ray.

The guidelines also require that health monitoring is applied to employees who work at a mine or mines for one month or a cumulative period not exceeding three months over a 12 month period. Further information on monitoring is provided in Biological Monitoring Guidelines by Department of Mines and Petroleum Resources (WA) (1997).

Coal industry employees’ health scheme

The Queensland Coal Board (1993 revised 1998) published an instruction manual to assist persons and organisations who had obligations within the health scheme. The Australian Industrial Relations Commission (AIRC, 2004) determined that when the instruction manual was referred to in an industrial agreement, it constituted part of the mines health and safety management system. Ham (2000) documented the evolution of the health scheme in some detail.

EXPOSURE STANDARDS


<table>
<thead>
<tr>
<th>Exposure</th>
<th>Guidelines</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cancer</td>
<td>La Dou (1994)</td>
</tr>
<tr>
<td>Diesel particulates</td>
<td>AIOH (2004)</td>
</tr>
<tr>
<td>Heat</td>
<td>AIOH (2003)</td>
</tr>
<tr>
<td>Whole body vibration</td>
<td>McPhee, Foster and Long (2001)</td>
</tr>
<tr>
<td>Commercial vehicle drivers</td>
<td>Austroads Inc (2003)</td>
</tr>
</tbody>
</table>

FREQUENCY OF MONITORING

Grantham (2001) examines monitoring strategies in relation to frequency of sampling and reliability of exposure estimates. There is elevated risk when the measured exposure is within 50 per cent of the exposure standard. He suggests that in this case, one sample per shift per ten workers should be undertaken each month.

An alternative approach was suggested by Ham (2002) who suggested that the frequency of sampling should be determined by both the need for accuracy in the determination of exposure and the amount of variation in the regular sampling program. Using the example of respirable dust monitoring, high variation in dust estimates was acceptable if the exposures found to be low, but for positions where exposures approached the statutory limits, more frequent sampling would be required to obtain a reliable estimate of cumulative dust exposure. Mines with high variations should undertake more sampling that mines where the range of exposure was relatively small.

COAL MINING COMPETENCIES AND TRAINING

In most jurisdictions, there are requirements for training programs to ensure workers and supervisors are competent to undertake their duties. In recognition of the need to upgrade standards, coal industry competencies (NTIS, 2005) have been developed recently in health and hygiene management systems as shown in Table 7. TAFE NSW with funding from Department of Education and Training developed numerous qualification guides, trainers’ guides and assessment guides including a trainers guide in ‘Implement and Monitor Health and Hygiene Management Systems’, see Table 8.
DICHOTOMY BETWEEN EXPOSURE STANDARDS AND SAFETY MANAGEMENT SYSTEMS

While personal protective equipment is required where exposure limits may be exceeded, a higher level of safety management is also required. Grantham (1994) and the Department of Natural Resources and Mines, Qld (2004) agree that this includes both health surveillance and enhanced training and supervision. There is little advice in how to manage the risk associated with moderate and high levels of exposures in a safety management/risk assessment framework except to say the health surveillance should be implemented.

In order to place the elevated exposures into a risk assessment and safety management framework, Ham (2004a) developed concepts for the following:

- comparable health outcome measures;
- definition of unacceptable health outcomes;
- measures for assessing the risk;
- trigger levels for various interventions in response to rising risk;
- development of interventions; and
- agreement between management and workers on the monitoring, the triggers and interventions.

HEALTH OUTCOME MEASURES

One of the obstacles in measuring, monitoring and focusing resources on improving occupational health outcomes is the failure to have a suitable benchmark parameter. The Global Burden of Disease approach discussed by Mathers, Vos and Stevenson (1999) draws on an international program that uses a unit called a ‘disability adjusted life year’ (DALY) as a common measure of harm caused by various diseases and injuries. This unit has two components – years of life lost (YLL) due to premature mortality plus the equivalent of healthy years of life lost due to disability (YLD). This provides a measure of comparing the human cost of life and quality of life lost due to mine explosions, motor vehicle accidents, stress disorders, cancers and hearing loss. In their study on the general population, cardiovascular disease and cancer were responsible for the highest years of life lost while mental disorders and the nervous system disorders caused the highest disability losses. The weightings per year for common mining related disorders are shown in Table 9.

The DALY is calculated as a loss from the group life expectancy. In 1996, the life expectancy for Australian males was 75.6 years. Mathers, Vos and Stevenson consider it pertinent to follow the overseas model and use a discount rate three per cent per year. For example a 56 year old who contracts dust related emphysema would lose \(20 \times 0.5\) years. After applying the discount factor, the net loss of 7.6 years.

This unit as developed to compare the impact of various disorders in a single population and to compare populations for the distribution of disease burden. Morfield (2004) discusses how the approach may be used to analyse the impact of a particular disease on a particular cohort in comparison to a control group. In this particular application, the burden of disease in the study...
The nature of data that can be extracted using this approach is that in the example, other causes is far the highest and warrants investigation. After that, trauma followed by heart disease would be key areas of focus while cancer and respiratory have a lower impact and would be secondary targets. The data suggests that miners mental health is better that expected in the general population. The mental health data contrasts with results found from the QCOS data in Table 3. This is an issue for future investigation.

**ASSESSMENT OF RISK**

The next challenge is to define the limit of what is an acceptable risk on occupational injury. At one level, it may be argued that no injury is acceptable. While this is commendable, a safety management systems based approach requires monitoring and this monitoring is designed to identify trigger levels that signify that some probability that an acceptable risk of harm has been exceeded. What is sought is a level of evidence that occupational exposure have resulted in a statistically significant variation from the normal range of human conditions in the un-exposed population.

Using a risk based approach, Donoghue (2001) suggests that an acceptable occupational probability of death of is $10^{-5}$ per year which is one tenth of the general community risk due to motor vehicle accidents. By combining this with the concept of years of life lost, an acceptable risk (probability times outcome) to a 25 year old who has a life expectancy to 75 years is $5 \times 10^{-4}$ years life lost. For example a 56 year old who contracts mild hearing loss would lose $(20 \times 0.01)$ years. After applying the discount factor, the net loss of 0.15 years lost quality adjusted. This is two orders of magnitude less that the standard suggested by Donoghue, but it is in line with the minimum payouts under the workers compensation arrangements.

The evidence indicating the risk of an unacceptable outcome may take a number of forms. Firstly, dose-response studies may be used to predict long-term outcomes and when the trigger health parameter level is reached, exposed persons should be withdrawn from the hazardous environment. A second approach is to use studies that monitor cumulative dose and assess these against expected final health outcomes to trigger some intervention. This is the approached used in radiation cancer related studies.

**MANAGEMENT OF UNACCEPTABLE HEALTH OUTCOMES**

The first step is to define what is an acceptable health outcome from and occupational health and safety perspective. If we work from the perspective that the mining industry should be free of occupational injury or disease (Department of Natural Resources and Mines, 2004), then it is acceptable if miners suffer injury and disease in line with community norms. This include freedom from work related disease and injury, but also includes the concept that life expectancy should not be reduced as a result of occupational injury or disease (Department of Natural Resources and Mines, 2004), then it is acceptable if miners suffer injury and disease in line with community norms. This include freedom from work related disease and injury, but also includes the concept that life expectancy should not be reduced as a result of occupational exposure (Rudd, 1998).

What is unacceptable then is predictable and preventable injury or disease that can be attributed to some element of the work method, work arrangement and work environment. Lost time injuries are usually considered to be work related. It is more difficult to establish that injuries suffered outside the work environment are work related and some statistical analysis of the events, exposures and injury is needed to establish that it is a work related disorder. It is plausible to associate fatigue related travel injuries and mental disorders to extended shifts and night work.

Prolonged exposure to coal and silica dust and fumes are known to be associated with various forms of respiratory disease. The most common are coal workers pneumoconiosis (CWP) and silicosis. There is an argument low rates of CWP and silicosis demonstrate the current dust management systems are effective.

The work on dose-response studies by deKlerk and Musk (1998),

---

**TABLE 9**
Disability weightings for healthy years of life lost.

<table>
<thead>
<tr>
<th>Weight</th>
<th>Disorder</th>
</tr>
</thead>
<tbody>
<tr>
<td>From</td>
<td>To</td>
</tr>
<tr>
<td>0.01</td>
<td>0.05</td>
</tr>
<tr>
<td>0.05</td>
<td>0.10</td>
</tr>
<tr>
<td>0.10</td>
<td>0.15</td>
</tr>
<tr>
<td>0.15</td>
<td>0.20</td>
</tr>
<tr>
<td>0.20</td>
<td>0.30</td>
</tr>
<tr>
<td>0.30</td>
<td>0.40</td>
</tr>
<tr>
<td>0.40</td>
<td>0.50</td>
</tr>
<tr>
<td>0.50</td>
<td>0.65</td>
</tr>
<tr>
<td>0.65</td>
<td>0.80</td>
</tr>
<tr>
<td>0.80</td>
<td>1.00</td>
</tr>
</tbody>
</table>

**TABLE 10**
Example of mortality study results (from Table 3).

<table>
<thead>
<tr>
<th>Cause of death</th>
<th>Control group</th>
<th>Exposed group</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>No</td>
<td>Median Age</td>
</tr>
<tr>
<td>Cancer</td>
<td>311</td>
<td>71</td>
</tr>
<tr>
<td>Heart disease</td>
<td>355</td>
<td>76</td>
</tr>
<tr>
<td>Mental disorders</td>
<td>29</td>
<td>42</td>
</tr>
<tr>
<td>Respiratory</td>
<td>64</td>
<td>77</td>
</tr>
<tr>
<td>Trauma</td>
<td>21</td>
<td>39</td>
</tr>
<tr>
<td>Other causes</td>
<td>220</td>
<td>84</td>
</tr>
<tr>
<td>All deaths</td>
<td>1000</td>
<td>74.4</td>
</tr>
</tbody>
</table>

The nature of data that can be extracted using this approach is that in the example, other causes is far the highest and warrants investigation. After that, trauma followed by heart disease would...
Coggan and Taylor (1998) and Rudd (1998) provide evidence that dusts significantly contribute to chronic bronchitis, emphysema and lung cancer that reduces life expectancy.

The problem of defining work related disease is more difficult when it comes to disorders which are common in the general community. Respiratory disease is common in the general population and is often fatal in the elderly. These disorders are exacerbated by the recreational habit of tobacco smoking. Smokers then are a higher risk group and thus it may be prudent to treat their respiratory disease risk in a different manner to the non-smoking population.

Mental disorders occur in the general population. The association of mental disorders with the work environment is more challenging and there are often few warning signs of early progression of potential serious and life threatening disorders. Guidelines on fatigue management (Department of Natural Resources and Mines, 2001) identify mental health issues such as stress, anxiety and depression as risks associated with night shifts and extended shifts but few management strategies are available to effectively manage these risks. The guideline notes that:

*Those already suffering from digestive disorders, diabetes, heart disease, psychological problems, alcohol and drug addictions and chronic sleep disturbances face additional burdens.*

It is possible that an effective health and safety management system should provide for special arrangements for the significant number of individuals who may fall into the above groups.

The issue is that occupational disease is no longer confined to strictly exposure related disorders but includes numerous common disorders that exhibit higher rates of incidence in the coal mining cohorts that in a non-exposed population. This conclusion has some fundamental implications for the design of health and hygiene management systems. Where a specific exposure may contribute to the development of a disorder, there is a case that a health surveillance program should collate data on the cumulative exposure and assess the risk of a related adverse outcome by comparing the cumulative exposure with known dose-response statistics. Furthermore, the program should also examine health indicators that provide early warning of a pending disorder.

For disorders where there are no reliable indicators of deterioration of health, there is a case that a trigger level based on cumulative exposure should be set based on past or future dose-response studies. Such trigger levels need to be set based in epidemiological studies that reliably determine that the rising risk of an adverse health outcome is predicted by increasing cumulative exposure. Workers, employers and regulators may set these levels by negotiation.

**CONCLUSIONS**

The detail of components of health and hygiene management systems are yet to be fully developed. The concept of combining health surveillance, cumulative exposure monitoring and analysis of health outcomes has merit as a basis for exposure based risk assessment and management but will be challenging to implement.

The notion of disability adjusted life-years lost provides a means of comparing short- and long-term occupational disorders of varying severity. When used cautiously and supported by good epidemiology, this process provides an effective measure of assessing and comparing disorders with long latency periods.

The emergence of new risks with long latency periods may be first indicated by subtle changes in mortality data. Several approaches have been demonstrated including age specific death rates, life years lost and proportions of fatalities. There are opportunities to better develop concepts of the application of trigger levels to change in health parameters in health surveillance and in cumulative exposure monitoring. The process of developing these concepts requires barriers due to confidentiality, discrimination and competitive short interests to be overcome. While many of these activities may be contacted out to researchers and health professionals, the complexity of mining OHS management systems is such that high level mining OHS professionals are needed to oversee data collection, analysis and setting and implementation of trigger levels. This issue is sensitive from an industrial relations perspective and a level of tripartite participation is necessary for settlement to be reached.

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Taking Stock of Safety

G L McDonald

ABSTRACT
My early experience with the coal industry was in the latter half of the 1970s, where attention to only ones own ‘accidents’ and concentration on behaviour controls were impeding the introduction of effective engineering controls such as ROPS, FOPS, roof support and residual current devices. Lack of the broad picture of personal damage and of relevant veridical (true saying) knowledge and data had to be overcome to achieve progress in reducing fatalities. Thirty years on, it is again necessary to take stock. How good is our knowledge, our data, our information on which we make decisions. Despite Australian all industry 1992-93 data showing 80.5 per cent of costs came from occurrences which permanently altered people’s lives, New South Wales is still the only state reporting permanent disabilities. In 1991 - 1992, the rate of permanent disability per 1000 wage and salary earners was two. In 2000 - 2001, it was over four – more than doubled. Also strongly motivating is the huge cost. Industry carried 25 per cent of the $34.3 billion costs for 2000 - 2001, up ten per cent of 82.8 billion if the costs of pain, suffering and early death are included.

INTRODUCTION
Community, industry, corporate and individual memory dim with time so that valuable knowledge and experience disappear and becomes unavailable. Stories help consolidate memory and make available lessons from the past. This in many ways is my story as a safety consultant for 30 years. In part it is the story of aspects of corporations; it began and was shaped in the coal industry, spread through other industries to finally contemplate the whole of Australian industry.

The quest, first dimly felt and later crystallised was the minimisation of the permanent alteration of people’s lives which accounts for 82 per cent of the total cost of personal damage from work. If pain, suffering and early death were costed in it would go over 90 per cent of total costs.

ROLLOVER PROTECTIVE STRUCTURES
Fresh from research on tractor ‘accidents’. study of mechanical engineering and psychology, and with a background of tractor driving, I was called by UTAH to Goonyella to investigate the overturning death of their most skilled front end loader operator while loading coal into a haul unit in the mid 1970s. By then rollover protective structures (ROPS) had over 15 years of established effectiveness of saving tractor drivers’ lives, since 1958 in Sweden. Mining folklore was strong. ‘Drop the bucket’ would go over 90 per cent of total costs.

Experience showed that destroying a person’s belief in the folklore without supplying a positive alternative provoked considerable anger. Convincing an operator that ‘dropping the bucket’ was not a reliable method of stopping overturning and then requiring the person to operate the machine without a ROPS was unacceptable. Anxiety is a difficult emotion to deal with and high anxiety levels can severely adversely affect a person’s performance. It was found that a person generally would not relinquish their belief in their particular folklore until the need for it to convince them they were not vulnerable no longer existed. For many years nurses were strongly trained in ‘correct’ methods of lifting. This training made them psychologically comfortable while they overloaded and permanently damaged their bodies, mainly the spine in the lower back. A ban on the manual lifting of patients has dramatically reduced such damage.

There is an element of truth in folklore, but it is a limited truth. Each work group would laugh heartily at other industry folklore but became disturbed and uncomfortable when their own beliefs were challenged. When one group was asked to identify their folklore, a supervisor summed it up well when he said:

The belief in ‘human error’ causation and a belief that only the operator’s experience was relevant delayed progress.

CLUTHA EXPERIENCE
In the late 1970s Clutha operated a number of underground coal mines in New South Wales. They had developed a safety program and needed a ‘motivator’ to get it to work. They had a lost time injury frequency rate (LTIFR) well up in the hundreds and over the past ten to 15 years had had nearly 30 fatalities, half from roof falls and half with machinery. Their safety target for the next year was a LTIFR of zero. This target was unrealistic and strongly indicated a lack of understanding. A great deal of underground work was under unsupported roof. They relied on ringing the roof (striking the sandstone roof with a metal rod). When asked by moving my hands apart vertically what thickness of sandstone was required to give a clear metallic ring rather than a thud, the group stopped the hands at about ten inches (250 mm). When given the proposition that a clear ring meant that either there was a good roof or that when a fall came it would be a big one, the group replied ‘it’s always the good roof that kills you’. More folklore.

THE FOLKLORE HURDLE
Experience showed that destroying a person’s belief in the folklore without supplying a positive alternative provoked considerable anger. Convincing an operator that ‘dropping the bucket’ was not a reliable method of stopping overturning and then requiring the person to operate the machine without a ROPS was unacceptable. Anxiety is a difficult emotion to deal with and high anxiety levels can severely adversely affect a person’s performance. It was found that a person generally would not relinquish their belief in their particular folklore until the need for it to convince them they were not vulnerable no longer existed. For many years nurses were strongly trained in ‘correct’ methods of lifting. This training made them psychologically comfortable while they overloaded and permanently damaged their bodies, mainly the spine in the lower back. A ban on the manual lifting of patients has dramatically reduced such damage.

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1. Geoff McDonald and Associates Pty Ltd, 36 Judith Street, Crestmead Qld 4132. Email: gmcdsafety@uq.net.au

whatever our folklore is we believe they are truths.
Someone from outside, who does not need to believe them, has to recognise our folklore for us.
COOPERATIVE APPROACH

The coal industry has moved a long way from those relatively recent times. It is not always known or remembered what has been effective. In 1978 UTAH’s Blackwater Mine imported the first of Euclid’s short wheel base 120 tonne prime mover to haul coal. The machine was stood down and operators refused to drive it. The importers asked for an evaluation of the ride to determine whether there was a problem with the prime mover or whether they were caught in a union-management disagreement. Both a ride in the unit and measurement of its ride vibration confirmed it as too rough to be driven. When the manufacturer was unable to modify the ride, the union and management agreed that the unit should not be put into service. This cooperation meant an unknown number of Australian miners have been saved from permanent alteration of their lives.

In early 1996 contact from the United States of America revealed that a number of these units had been sold into ‘green field’ sites where the operators (sons of local farmers) had no haul driving experience for comparison. A large number of operators suffered vertebral end-plate fracture in the thoracic spine and in some, damage continued into the lumbar spine. The unit ‘lapped’ or ‘pitched’ with the vibration varying from 40" to 50" to the horizontal. The horizontal accelerations equalled the vertical. To stabilise the spine the back muscles operating at a very shallow angle to the spine had to use maximum or near maximum contraction to prevent the upper body rotating forward over the pelvis. This strong contraction placed excessive compression loads in the spine which led to the damage.

SLOW ADOPTION OF EFFECTIVE ENGINEERING INNOVATION

During this era a sad chapter in Australian safety history was being played out. Australia was largely oblivious. In 1969, this author had an earth leakage circuit breaker (ELCB) fitted to all circuits of his house being built. In the same year a Japanese labour safety and health regulation required the fitment of ‘differential current operated protective devices’ in the manufacturing industries. In that year (1969) there were 39 deaths, by 1972 there were 18, in 1980 there was one (Whiteman, 1987).

During this time Australian authorities, including Standards Australia, were working strongly on ‘rules’ which everyone concerned had to follow. Consulting to a Regional Electricity Board in the early 1980s led to the insight that authorities were worried that fitment of ELCBs would encourage people to be less careful, ie not follow the rules as well. It was many more years before there was strong encouragement to fit ELCBs. Rules had more influence on what was done than did the knowledge of the devices effectiveness in Japan.

EXAMINING THE EVIDENCE

Taxonomic classification of past damaging occurrences according to types of energy used was developed in the coal industry and its application in the petrochemical industry produced new understanding. Records of 1037 cases of personal damage in one organisation showed when by reporting to an Industry Commission’s 1995 report, New South Wales is still the only state reporting permanent disabilities. All other states are unaccountable for 80 per cent of the total cost of damage to members of the community on whose behalf they are required to manage work health and safety.

The National Occupational Health and Safety Commission (2002) (NOHSC) set ten year targets as follows: sustain a significant, continual reduction in the incidence of work-related fatalities with a reduction of at least 20 per cent by 30 June 2012, and a reduction of ten per cent being achieved by 30 June 2007, and … reduce the incidence of workplace injury by at least 40 per cent by 30 June 2012 (with a reduction of 20 per cent being achieved by 30 June 2007).

No mention is made of reducing cases which permanently alter people’s lives non-fatally, ie those cases which account for 80.5 per cent of the cost in 1992 - 1993.

To examine change over recent years, it is necessary to look to New South Wales WorkCover since they are the only source which includes information on permanent disability. New South Wales WorkCover figures have been plotted and lines of regression calculated for the years 1991 - 1992 to 2000 - 2001. From the line of regression ‘all occurrences’ have declined from 28 per 1000 wage and salary earners in 1991 - 1992 to 22 in 2000 - 2001, a ten year change rate (decrease) of 22.7 per cent as shown in Figure 1.

By contrast, as shown in Figure 2, ‘permanent injury’ has increased from two per 1000 wage and salary earners in 1991 - 1992 to around 4.4 in 2000 - 2001. The rate of change is equivalent to an increase of 140 per cent over a ten year period.
On the basis of these figures, the national targets set by NOHSC could be met, indicating desirable progress had been made when the quantity of damage had actually increased.

Further confirmation of the increasing quantity of damage in the Class I region comes from the National Occupational Health and Safety Commission’s (2004) updating of costs of work related injury and illness to the year 2000 - 2001. Whereas 82 per cent of costs in 1992 - 1993 came from 13 per cent of occurrences, 92 per cent of costs came from 15 per cent of occurrences in 2000 - 2001. Again these costs do not include a consideration for pain, suffering and early death and totalled $34.3 billion. A further costing for pain, suffering and early death was made and brought the total cost from work damage to people to $82.8 billion. When the cost of pain, suffering and early death is included, 15 per cent of occurrences account for 96.5 per cent of costs.

This is powerful and surely irrefutable evidence that effort must be directed to the top end of the severity scale – Class I damage.

McDonald E L (1997) produced a taxonomy, ‘Accidents in the Coal Mining Industry 1990 - 1995’ for the New South Wales Minerals Council and the Queensland Mining Council. This sought to describe the origins of Class I damage but functionally had to settle for dealing with cases where greater than 90 days work was lost and fatalities. Eight hundred and ninety two underground mining and 312 open cut mining cases were classified according to the type of energy. For all coal mining human (39.8), gravitational (32.3) and machine (20.6) accounted for 92.7 per cent of occurrences, while human (38), gravitational (34.7) and machine (22.09) accounted for 94.8 per cent of costs.
Careful study of the taxonomy gives interesting insight. In considering open cut mining, combining a number of different taxa gives a picture of ‘ride disturbance’ which accounted for 27 per cent of occurrences and 49 per cent of cost. Figure 3 shows 74 cases occurred while the vehicle was moving and most involved vibration and jarring, with four coming from the operator impacting the vehicle interior. The ‘stationary’ cases mostly involved loading. This taxonomy provides a platform for a more detailed assessment which would identify such problems as mismatching of the ride characteristics of a machine and of the vibration isolation characteristics of the installed seat, as well as operational problems.

Over recent years attention has been directed to the Australia wide problem. Each day in 1992 - 1993, 137 people working in Australia had their life permanently altered by damage from work – 50 000 people a year. Of these approximately 20 000 will never work again and 30 000 work for fewer hours or on less skilled work. By now many more people per day will have their lives permanently altered. A nation cannot afford this loss.

If the late 1970s early 1980s experience with the coal and other industries is interrogated, it shows specific issues related to death were identified, specific solutions were developed and adopted. This occurred within a framework of the development of a new understanding and resulted in the gradual rejection of only partially true folklore. Its anxiety managing influence became unnecessary. WorkCover (1998 - 1999 – 2000 - 2001) shows that the three largest defined groups ‘manual handling’ 35 per cent, ‘falls to the same level’ 23 per cent and ‘falls from height’ 11 per cent account for almost 60 per cent of cases. This appears different from human (39.8), gravitational (32.3) and machine (20.6) for McDonald’s coal taxonomy until it is realised that the vast majority of WorkCover 15 per cent ‘other’ involves motor vehicles. On this basis the WorkCover data can be referenced human (35), gravitational (24) and machine (15). Other cases would fit into these three groups so that the overall energy sources would be similar to that of the coal taxonomy. In many cases, however, the details of what happened in each case will be vastly different. It is knowledge of this specific detail which is critical, not only of individual cases but also of aggregated similar cases, to show clearly ‘common pattern’ and ‘individual differences’ so that specific countermeasures can be developed on the basis of veridical (true saying) knowledge.

**THE SAFETY PARADOX**

The safety paradox must be confronted. On the one hand there are far too many Class I occurrences, on the other hand each specific type of Class I damaging occurrence is so rare that an individual who has to make safety judgements has no experience (first, second or third hand) or knowledge of them. They may well know the applicable folklore. Millions of person years of experience have to be aggregated to compile the appropriate detailed knowledge. The coal taxonomy was a start to aggregating and presenting information. It needs to develop further and grow on a continuous basis.

Our governments have the responsibility of managing work health and safety on behalf of the community they represent. They have adopted Robens style legislation which strongly requires management-workforce consultation and appears to assume the workforce knows what is required. In many cases they do not. The knowledge of what is required is embedded in the 137 a day (50 000 a year) cases of permanent life alteration. Sufficiently detailed case histories of these cases have to be aggregated, taxonomised, dissected and digested so that appropriate knowledge can be educated back to the workplace. While it is not clear who could most effectively do this, it is clear
that it is the government’s responsibility to see that it is done so that safety is not directed by lack of knowledge and by risk assessment necessarily based on the ‘feeling function’, as information is not available to enable the ‘thinking function’ to be used.

Within governments the safety function is under funded, with funds coming from industry and not from community generated money. According to NOHSC (2004), if pain, suffering and early death are not included in costing, the employer carries 25 per cent of the cost. Therefore the individual and the community carry 75 per cent of the cost. If pain, suffering and early death are included the employer carries 12 per cent and the individual and the community carry 88 per cent of the cost. The community has far more to gain by a reduction in Class I damage than does the employer. The employer has much to gain as better quality information gives more efficient and effective control effort.

Earlier it was stated that the total cost of work personal damage for 2000 -2001 was $34.2 billion without pain, suffering and early death being costing and $88.2 billion if it was. By way of illustrating the size of that amount it is noted that BHP Billiton was quoted during the proposed WMC take over as having a value of $89 billion. The cost of Australia’s work damage to people for one year is of the same order of magnitude.

CONCLUSION

The argument for focusing on Class I damaging occurrences (the 13 per cent) is very powerful as long as we know what they are. In mining you do not dig out all the material so that in the process you get the product (coal, iron ore, etc) you want. Rather, you map the formation, and in the case of coal, identify the seams and target them. Safety should work the same way.

The moment permanent alteration of life or permanent disability is mentioned the possibility (or in some people’s minds, the certainty) of malingering and fraud is raised. Nothing activates safer more. The prominence of, and our need to focus on Class I is robust.

The robustness is illustrated by testing the assumption that half of the Class I occurrences are not genuine. This gives 7.5 units of Class I damage and 85 units of Class II damage for a total of 92.5 units. Class I occurrences then make up eight per cent of occurrences. For costs with pain, suffering and early death included, the 96.5 halves to 48.25 units for Class I cost plus 3.5 units of cost for Class II, giving 51.75 units of total cost. Class I cost now is 93 per cent of total cost. Said simply, halving the number of Class I occurrences reduces Class I cost from 96.5 to 93 per cent of the total cost. Figure 4 graphically illustrates the above figures and shows that Class I occurrences will dominate the overall cost until very large Class I reductions have been made.

The development of a sufficiently detailed, comprehensive and organised information base to guide on site safety activities is necessary. This needs to include Class I injuries with detailed reporting of circumstances, damage and proposed preventative measures. The reporting of incidents which have potential to cause Class I injuries should be reported in the same detail. The data collection system also needs to look harder to capture data on the cases of Class I occupational disease and traumatic injuries that may have been contributed to by occupational exposures and disorders.

The state governments require that a vast number of Australians to identify hazards, do risk assessments and take effective control action – and get it right. If they do not they will be punished. This is equivalent to a teacher setting a class a subject, providing very limited resource material and examining with relatively obscure penetrating questions, with dire consequences for poor performance.

What do we need to do to develop a focusing database so that the upward Australian trend in permanent disability can be turned downwards?

REFERENCES


Legislation, Litigation and Liability

A J See

ABSTRACT

Safety and health at work is about people. It is about people who plan for work, organise work and perform work. It is about those who come in contact with the workplace and those who are responsible for the workplace.

There are wider ranging safety and health obligations that are imposed on the industry stakeholders that span all phases of the production process. In all cases, those responsible for discharging the obligations must ask the questions: Is what we propose safe?, can it be done safer? and are we sure?

What we learn from recent legal cases before the courts is that it is essential that all stakeholders work together in a collaborative way when discharging these obligations.

Safety and health at work cannot become an issue that is viewed only through the eyes of a single stakeholder group. The reason for this is quite simple. The interrelatedness of the responsibilities that are imposed on each of the stakeholder groups, means that the performance based legislation simply wont work, where an active model of communication and consultation is not in place.

INTRODUCTION

Australian workplace health and safety law has been the subject of much debate over the past several decades. In the late 1980s and early 1990s, many of the state based mining and non-mining laws were recast with a view to making health and safety a fundamental part of the production process.

Often the language of those involved in that debate spoke of pre-Robens versus post Robens legislation, with the predominant view being that performance based legislation in the style advocated by Lord Robens (Safety and Health at Work, 1972), was the panacea for ensuring health and safety at work. That is, what was required was legislating for outcomes, not prescribing the way in which safety was to be achieved.

But performance based legislation may not, in itself, be always that easy to implement.

While Australia’s safety and health laws are arguably among the best in the world, there nonetheless remains a good deal of debate as to how the obligations of the individual stakeholders should be carried out. This paper seeks to highlight some of the issues that are pertinent to the debate, through an examination of some recent case law. The analysis takes place not for the purposes of examining the behaviour of the parties, but more to gain a better insight into the nature of the issues that form the backdrop to the potential legislation, litigation and liability that faces the coal mine operator.

THE LEGISLATIVE BACKDROP

Understanding the obligations of the coal operator and the site senior executive

The starting point for any analysis of this type must be the current legislative backdrop and what I intend to do is to consider this backdrop from a Queensland’s perspective.

The Queensland Coal Mining Safety and Health Act 1999 has two objectives, to protect the safety and health of persons at coal mines and persons who may be affected by coal mining operations and to require that the risk of injury or illness to any person resulting from coal mining operations be at an acceptable level (Section 6).

The primary obligations imposed on the coal mine operator focus on the place of work, the plant and the systems and people and it is through the role of the site senior executive that these obligations are discharged.

The most critical activity for the site senior executive is to ensure on behalf of the operator, that it develops and implements a safety and health management system for the mine and that it is supported by a management structure that will ensure that the system works.

The importance of having an effective management structure is made quite clear when one reflects on some of what the safety and health management system needs to do. For example, the system (Section 62) must:

- define the coal mine operator’s safety and health policy;
- contain a plan to implement the coal mine operator’s safety and health policy;
- state how the coal mine operator intends to develop the capabilities and support mechanisms necessary to achieve the policy; and
- include principal hazard management plans and standard operating provisions.

Yet the management structure of the operating company is not alone when it comes to the effective implementation of some of these responsibilities. Under Queensland law, the legislature has placed significant importance on the tri-partite responsibilities of all of the industry partners and this is evident in the roles given to the Safety and Health Council (Part 6), the Industry (Part 7) and Site (Part 8) Health and Safety Representatives.

The point to be made is that the legislative arrangements for the coal mine operator are complex and involve many stakeholders.

The following two case studies have been selected to demonstrate these complexities.

A CONSIDERATION OF THE OBLIGATIONS IN THE CASE OF GRETLEY

Background

On 14 November 1996, four mine workers at the Gretley Colliery were killed, when the continuous miner they were operating holed into abandoned workings of the Young Wallsend Colliery (YWC) causing a sudden inrush of water. Such was the force of the inrush that the 45 tonne continuous miner was moved 20 metres inbye and found after the accident, positioned diagonally across the heading.

A central factor in the disaster was that the south east boundaries of the old workings were always 100 metres or more closer to the proposed mining activity boundaries for 5051 panel than the official mine plans for the colliery were depicting at all relevant times.

As a result of the fatalities and following a judicial inquiry before the Court of Coal Mines Regulation and a coroner’s inquest, 52 charges were brought against the corporate defendants, the Newcastle Wallsend Coal Company Pty Limited

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Let us consider these obligations within the context of the facts of the case.

The record tracings

At issue in relation to the planning, research and assessment charges was the reliability of the record tracings that were held by the Department of Mineral Resources. The record tracings (RT 523) were made up of three sheets.

The first sheet, Sheet 1, was headed ‘Plan Showing Young Wallsend Coal Workings’ and was copied from the colliery plan at the coal field office in 1892. Sheet 1 contained two sets of workings that were depicted separately by black and red ink. According to Staunton J (IRC, 2004), the two sets of workings appear to overlay each other, particularly in the north-western and south-eastern boundaries, so much so that:

any person looking at RT 523 Sheet 1 could not help but wonder as to the precise import of the red and black workings and their relationship to each other (at IRC, 2004, page 388).

By contrast, Sheets 2 and 3 came into existence some time around 1980 and were created according to a departmental Minute Paper written at the time:

due to the fact that the workings of both seams that directly overlay each other and were shown by differing colours on the one plan of abandonment, as well as the poor condition of the plan, that a separate tracing of each working had to be made (at IRC, 2004, page 389).

The purpose of Sheets 2 and 3 was to separate the red and black mine workings depicted on Sheet 1, with the bottom seam workings appearing to be represented in Sheet 2 and the top seam workings being reproduced in Sheet 3. The problem that appeared to arise as a consequence of the reproductions of these workings was that:

whoever created them interpreted Sheet 1 in a particular way. That is, the red and black workings depicted as superimposed on each other in Sheet 1 had been separated out and depicted as stand alone workings in two different seams, vertically 18 metres apart (at IRC, 2004, page 401).

The inaccuracy of these maps proved fatal to the company and it was the preferred view of at least one witness that it was more than likely that the drawings related to the workings of the upper seam only. According to Staunton J, the defendants failed to do research and planning properly, because they failed to independently and objectively consider anomalies in Sheet 1 that Sheets 2 and 3 didn’t resolve.

In this respect there are several issues that should be noted. Firstly the court accepted the evidence of an expert witness that there were basic surveying principles ignored by the surveyor, when he was confronted with the glaring inconsistencies identified in Sheets 2 and 3.

Secondly, the court was of the view that against that backdrop, the defendants should have sought additional information from the department, such as from the Abandonment Register to clarify the extent of the workings. There was no evidence of this taking place.

Finally, an independent drilling program undertaken by the defendant established that the purported Borehole Seam workings in the south-eastern direction as depicted in Sheet 2, did not exist as had been depicted.
To summarise the court held that the:

Defendants did not critically scrutinise Sheets 2 and 3 seeking background information to satisfy themselves about that information causing Sheets 2 and 3 to be depicted the way they were. Furthermore, no filed notes were made of surveys, and no abandonment plan of YWC was filed; therefore there was no certainty of the extent as to the continuation of old workings. This was actually confirmed by an independent drilling report throwing doubt onto the whole situation.

It was the court’s view that the defendant’s reliance on Sheets 2 and 3 as the basis for planning mining at Gretley, especially in 50/51 panel created a real and potential risk to the health and safety of its employees working in that panel.

**Further derivative planning and research charges**

What is important to observe in the Gretley analysis, is the reliance by the Crown prosecutor on derivative charges. That is, further charges that are brought about derived from an initial alleged failure. While the court recognised that the prosecutor was technically correct in casting the charges this way, His Honour was of the view that such a process could on occasions be unnecessarily duplicitous.

That being said, the court held the corporate defendants liable for the following derivative charges stemming from their failure to undertake planning and research:

1. the defendant’s failure to accurately depict the location/extent of YWC old workings on any mine plans;
2. the defendant’s failure to accurately depict the location/extent of YWC old workings on the Application submitted to the Department on the 6 September 1994;
3. the defendant’s failure to accurately depict the location/extent of YWC old workings on the redrawn plans forwarded to the Department on the 27 October 1994; and
4. the defendant’s failure to accurately depict the location/extent of the YWC old workings on the Variation submitted to the Department on or about 11 August 1995.

*The court dealt with these derivative charges together and found the defendant’s guilty on each occasion, because once established that the defendant failed to accurately depict the YWC old workings on the initial mine plan of Gretley, they would continue to do so in all future mine plans subsequently produced by Gretley.*

**Failure to undertake appropriate risk assessment**

Another example of planning charges that were dealt with by the Court related to:

- a failure to plan by way of risk assessment for the development of 50/51 panel; and
- a failure to carry out an assessment of the risks to the health, safety and welfare of the employees and mine workers in the event of an inrush of water and/or dangerous gases.

Despite the contention of the corporate defendants that a risk assessment process had taken place by the companies when the original minivall application was made, the Court was of the view that such a process was not the same as a documented risk identification and assessment process that would include a risk management policy; duties and responsibilities of persons involved, a risk register and risk action plan.

His Honour stated:

Given the nature of the risk, an adequate risk assessment would have encompassed much more than acknowledging the presence of the old workings and the intention to leave a barrier. In identifying risks as being the risk of inrush from water and/or dangerous gases, the consequences of such a risk would have been identified as death or injury to workers. This would have highlighted as a risk prevention strategy the need to ensure that the depiction of the Young Wallsend old workings could be relied upon with question as to their accuracy (at 550).

The Court (IRC, 2004) found that while the Gretley Collieries Emergency Procedure Document identified clear procedural steps to be followed at an administrative level once the incident leading to a decision to evacuate had occurred, there was nothing in the document that directed the actual employees at the site of the major incident.

The observation was also made by the Court that there was no direct evidence received from any witness who worked at the Gretley mine, as to their knowledge of and reliance upon the Emergency Procedure Document.

To summarise the above, the scope of the planning charges were wide ranging and impacted on all aspects of the health and safety system.

**System of work charges**

The second type of charges laid against the corporate defendants, were the system of work charges.

Again while the nature of these charges have as their foundation the initial failure by the defendants to undertake effective planning and research, the analysis of the issues did identify several unique considerations.

Of interest are those aspects of the charges that are particularised to include:

- a failure to investigate, adequately or at all, Deputies written and oral reports from 1 November 1996;
- a failure to inform Deputies, the employees and other mine workers that the Young Wallsend coal workings were full or water and under a head of pressure; and
- a failure to instruct Deputies, the employees and other mine workers to be vigilant in looking for signs of water make whilst working in the panel.

It is worth noting that on three occasions before the accident, that the Mine Deputy had entered into his statutory reports the presence of water in the 50/51 panel.

These included:

- 1 November 1996 – nuisance accumulation of water;
- 4 November 1996 – large amount of nuisance water; and
- 13 November 1996 – coal seam is giving out considerable amount of water seepage at face of C heading.

While the court did not conclude that these reports were extraordinary in themselves, coupled with the fact that the defendants were relying on inaccurate mine plans created a situation of far more significance.

The court also heard evidence of a discussion held between the mine surveyor and the Mines Subsidence Board several weeks before the accident, when the surveyor was advised that:

> we were having a water management problem and management wanted to know where these plans were or the accuracy of the plans.
Additional evidence was given as to the presence of a contractor’s hydraulic drill rig that may have been brought in for drilling ahead in the 50/51 panel either scheduled for the day or the following day of the disaster.

The conclusions of the Court in relation to these charges were that the system of work charge is derivative in nature as it stems directly from the defendants’ failure to properly research and assess the location and extent of the YWC old workings.

The Court held that although the mine workers knew that they were working towards old workings where they thought they were, the court held that the mine managers and workers should have been made fully aware of the YWC old workings. As a consequence, the defendants were found guilty of the system of work charges.

Night shift charges

The final category of charges related to the time period for the night shift of 11.30 pm 13 November 1996 until 7.30 am on 14 November 1996.

The night shift charges were particularised in the exact same terms as the system of work charges.

Again in the case of the night shift charges, the majority of the particularised failures alleged, derived from the defendant’s primary failure to properly research the location and extent of the Young Wallsend old workings.

While according to the Court these charges relied on differing factual particulars or differing aspect of primary fact in order to establish the basis of the alleged offence, for predominantly the same reasons and conclusions the majority of the failures as particularised were proven.

Defences under Section 53 of the OHS Act

Did the defendants do all that was reasonably practicable?

A defence under Section 53(a) requires the defendant to meet the objective test as to whether it was reasonably practicable for the defendant to have complied with the Act. In WorkCover Authority of NSW (Inspector Byer) v Cleary Bros (Bombo) Pty Ltd (2001) 110 IR 182 at 204, Walton J said that in assessing the merits of whether a defendant had done all that is reasonably practicable, requires a:

balancing of the nature, likelihood and gravity of the risk to safety occasioning the offence with the costs, difficulty and trouble necessary to avert the risk.

In that respect the corporate defence failed.

Staunton J found that it was always reasonably practicable for the defendants to:

- ensure that there was an adequate barrier between where the employees were working and the Young Wallsend coal workings;
- test drill, or cause test drilling to be performed to locate Young Wallsend coal workings;
- inform deputies, the employees and other mine workers that 50/51 panel was heading towards the Young Wallsend coal workings;
- inform deputies, the employees and other mine workers that the Young Wallsend coal workings were full of water and under a head of pressure; and
- instruct deputies, the employees and other mine workers to be vigilant in looking for signs of water make whilst working in 50/51 panel.

Defence that the corporation had no control

The second prong of the defence at Section 53(b) provides that it shall be a defence to any proceedings for the person to prove that:

(b) the commission of the offence was due to causes over which the person had no control and against the happening of which it was impracticable for the person to make provision.

The defendants sought a defence under this head on the basis that the existence and availability of information on the location and extent of the Young Wallsend Coal workings was not within the control of the defendants.

The court rejected that submission. It held that with the exception of one file, all other relevant information that went to researching the location and extent of the Young Wallsend Coal old workings was within the control of the defendants, in that it was readily accessible by them. Firstly, the Court observed that the defendants had the resources and personnel to enable them to carry out that task. Secondly, however, the Court was of the view that while it was correct for the defendants to assert that the errors made by the Department of Mineral Resources were not under the control of the defendants, that such a view missed the point.

The defendants were not being held liable for the errors made by the department, but for their failure to properly research the location and extent of the workings and the consequences that flowed as a result.

Individual defendant’s liability

The decision in Gretley has caused some degree of consternation among those persons engaged within the mining industry who hold statutory appointments.

The actions commenced against the eight personal defendants comes about by virtue of Section 50(1) of the Occupational Health and Safety Act 1983, that states:

(1) Where a corporation contravenes, whether by act or omission any provision of this Act or the regulations, each director of the corporation, and each person concerned in the management of the corporation, shall be deemed to have contravened the same provision unless he or she satisfies the court that...

(b) he or she was not in a position to influence the conduct of the corporation in relation to its contravention of the provision, or
were undertaken at the mine.

The central issue for the court to determine was whether the individual defendants were concerned with the management of the corporation beyond a reasonable doubt, and if so did they exercise due diligence to prevent contravention of the Act.

It was the view of Staunton J, that the decision-making authority and the inherent responsibility of the employee must affect the whole corporation or a substantial part of the corporation, for the employee to be ‘concerned in the management of the corporation’. There must be more than participation in the activities relevant to the responsibility and work undertaken at the mine.

In other words for an employee to be ‘concerned in the management of the corporation’ their responsibilities must go beyond the statutory functions under the Coal Mining Regulation Act.

Were the personal defendants concerned in the management of the corporation?

In the case of the Gretley mine manager he was appointed as a general mine manager as well as a statutory mine manager. The significance of this according to the court, was the corporate title intended to encompass a broader range of duties associated with the total management of the business, far more than just the statutory function.

For example, there was evidence of the general mine managers’ roles in the total management of the corporation. He attended and participated in meetings within the broader management structure of the corporate parent Oakbridge, and as part of that role was involved in the marketing, financial, direction and policy decisions of the corporate parent.

In addition, the Court heard that the general mine managers at Gretley implemented and oversaw the corporate safety meetings of Oakbridge, forming complex safety policies at various Oakbridge mines, including Gretley.

All of these responsibilities that were inherent in the role of the general mine manager, provided evidence that the position was one that was concerned with the management of the corporation. Yet that in itself is insufficient to establish a personal defendant. To do so requires a personal responsibility in the management of the corporation that includes some practical connection with the causal act or omission of the corporation.

In the case of the general mine managers, their practical connection with the corporate defendant’s failure to properly research the location and extent of the YWC old workings was established when they had signed applications for miniwall approval to the Department of Mineral Resources, where the application mentioned that the Borehole Seam workings are ‘full of water’ and ‘don’t pose a danger to Gretley workings.

Role of surveyor

In determining whether the Gretley mine surveyor was concerned with the management of the corporation, the court considered the duties of the surveyor as contained in Clause 8 of the Coal Mines Regulation (Survey and Plan) Regulation 1984.

Particularly relevant were the duties located at subclauses (c), (f) and (g) as follows:

(c) he or she, being in such a position used all due diligence to prevent the contravention by the corporation.

The personal defendant. To do so requires a personal responsibility in the management of the corporation. Yet that in itself is insufficient to establish a personal defendant. To do so requires a personal responsibility in the management of the corporation that includes some practical connection with the causal act or omission of the corporation.

In terms of the first tranche of the management test, the court determined that the mine surveyor had certified the accuracy of the incorrect mine plans. These plans were then utilised in supporting decisions taken at management level of the organisation to depict proposed future mining activity and the extent of current workings, and workings that have been abandoned.

This was sufficient for the court to determine that Gretley’s mine surveyor was concerned in the management of the corporation. The Court was also of the view that the mine surveyor had a practical connection, through decision making and advice, between the corporate defendants and the primary failure of the corporate defendants to properly research the location and extent of the YWC old workings.

Role of under manager

In determining whether Gretley’s under managers were concerned with the management of the corporation, the court looked to the statutory responsibilities of the under manager as contained within Section 41 of the Coal Mine Regulation Act 1982.

There was no evidence of any delegation of managerial responsibility by the mine manager to the under managers in accordance with Section 56 of the Act. In addition, Clause 9 of the Coal Mines Regulation (Managers and Officials – Underground Mines) Regulation 1984 causes the under managers to be responsible for mine safety only to the extent of the under managers’ jurisdiction. On that basis, the under managers were not held personally liable.

Due diligence required to avoid contravention of Section 50

The second prong of determining the case against the personal defendants once found to be concerned with the management of the corporation, requires an examination of the due diligence employed by those persons.

The case of the mine manager

In the case against the mine managers, the court held that evidence indicates they were in a position to influence the conduct of corporations regarding the contraventions of NWCC and Oakbridge already established under s15(1) and s16(1) of the Occupational Health and Safety Act 1983.

It was established that the mine managers did not use all due diligence to prevent contravention of the Act by either NWCC or
Oakbridge. The mine managers signed and approved plans and applications regarding approved plans of the previous mine surveyor of Gretley when the original applications were made to the DMR in 1994 and 1995.

These plans made by the previous mine surveyor incorrectly depicted on Sheet 3 the presumed old workings of YWC. Therefore the court held that the mine managers failed to discharge their statutory responsibilities under Section 37 of the Act and Part 3 of the Coal Mines Regulation (Methods and Systems of Working – Underground Mines) Regulation 1984.

In addition, the mine managers advised and were involved in corporate decision making through their participation in Oakbridge’s corporate strategic planning meetings regarding proposed mining activities at Gretley.

The Court found that Section 37 (2)(h) of the Act is quite clear with respect to the statutory responsibilities of the mine manager. That person must ensure that he or she possesses all available information regarding surroundings, the actual mine, and safety. The court held that it was clear that as the under managers at Gretley did not know of the extent of the YWC old workings and the potential for inrush of water, that the mine managers must have failed in meeting their obligations under the Act.

On this basis, Staunton J held that the mine managers did not exercise due diligence to prevent the contraventions by NWCC and Oakbridge occurring.

The case of the Surveyor

In assessing whether the mine surveyor exercise all due diligence, his Honour stated that by certifying the accuracy of mine plans relevant to Gretley, that the mine surveyor took on the liability this invites. According to Staunton J, it was clear that the mine surveyor did not use all due diligence to research the correct location and extent of the YWC old workings and the potential for inrush of water, that the mine managers must have failed in meeting their obligations under the Act.

The case for the applicant was that the two shafts at the mine, while trafficable entrances and thereby escapeways, were not adequately separated because a reasonably foreseeable event happening in one of the escapeways could affect the ability of persons to escape through the other escapeway.

THE GRASSTREES DECISION: A QUESTION OF INTERPRETATION

By way of contrast and in a much more concise way, I wish to turn to the issues raised in the case of Construction, Forestry, Mining and Energy Union v State of Queensland and AngloCoal (Grasstrees Management) Pty Ltd.

The central issue in the Grasstrees’ dispute was whether the underground mine required a third shaft in order to comply with Section 296(1) of the Coal Mining Safety and Health Regulation 2001 (Qld).

Section 296(1) provides:

The site senior executive must ensure the mine has

The applicant’s case was that the two shafts at the mine, while trafficable entrances and thereby escapeways, were not adequately separated because a reasonably foreseeable event happening in one of the escapeways could affect the ability of person to escape through the other escapeway.

Put simply, did the two present escapeways, one of which was a ventilation shaft only, have to be separated by a third escapeway for health and safety purposes in the event of one of the escapeways being unavailable.

Determining an acceptable level of risk

In considering this question, the Supreme Court of Queensland turned to a number of fundamental principles. Firstly, what is an acceptable risk of injury?

Section 29(2) of the Coal Mining Safety and Health Act 1999 sets out what is an acceptable level of risk from a mining operation.

In determining whether risk is within acceptable limits, regard must be had to:

a. the likelihood of injury to a person rising out of the risk; and
b. the severity of injury or illness.

At the heart of this issue, was the applicant’s concern of the incapacity of the existing system in the case of fire. That is, that a fire in the intake shaft would contaminate the air by producing smoke, reducing the oxygen content of the air and producing carbon monoxide, with the result being that the contaminated air would inevitably flow through the roadways and the exhaust shaft.

In assessing this issue, the Court turned to Section 37 of the Regulation that deals with the requirements for a coal mine’s safety and health management system, including issues relating to fire prevention and control, as well as the standard operating procedures. The legislation makes it clear that a risk of injury being at an acceptable level is dependant on the risk’s likelihood and severity.

The case for the applicant was that if the two escapeways in a mine are not adequately separated, in the event of a fire, the likelihood of injury to a person arising out of that risk, and the severity of that injury, are very real.

Lessons learnt from Gretley thus far

The ultimate result of Gretley is unlikely to be known for some time. What is clear though from this potted analysis of the issues is that the expansive nature of the safety and health responsibilities under the coal mining safety and health law cannot be understated. The implications for the coal mine operator and the statutory delegates such as the site senior executive are significant.

Consider for example, the implications of the planning activity, the reliance on departmental supplied mining plans and the spiralling effect that the inaccuracies of the mine plans produced on this occasion. Of significance also in this case, is the impact on the individual statutory appointee, particularly where that person is concerned with the management of the corporation.

While no doubt the further appeal against this decision is likely to produce a similar smorgasbord of issues to consider, the lesson to be learnt by all stakeholders is the sheer extent and significance of the mining safety and health management system.
Determining a reasonably foreseeable event

The real issue for the court to determine was whether the possibility of a fire down the Grasstrees’ mine shaft was a reasonably foreseeable event.

McMurdo J held that whether a potential event is an unacceptable risk depends on the nature and effectiveness of relevant controls. He said that a fire is an event, but no unreasonable risk is assumed if recommended controls are implemented.

It was his Honour’s view that whether a reasonably foreseeable event is a risk is a subjective test.

In this regard the court scrutinised the specialist evidence from the Minerals Industry, Safety and Health Centre at the University of Queensland.

Despite the fact that the court heard evidence that 2000 litres of fuel burning within the fuel pod would take some 280 minutes to completely combust and that such a fire would not adversely affect the ability of persons to exit the mine via the exhaust shaft, the court nonetheless was of the view that such a risk of injury for workers at the mine, was foreseeable.

In reaching his view, his Honour considers the application of the High Court (1980) decision in Wyong Shire Council v Shirt (1980) 146 CLR 40.

Applying the test in Wyong Shire Council

In that case, the High Court found that:

> When we speak of a risk of injury as being foreseeable, we are not making any statement as to the probability or improbability of its occurrence, save that we are implicitly asserting the risk is not one that is far fetched or fanciful (at 42).

Despite the fact that the company’s evidence was that the ignition of fuels within or split from the fuel pod could occur only through a combination of several human and/or mechanical failures, the Court decided that the event of a fire remained a real possibility unless safety mechanisms such as welding were regarded as incapable of failure through human error.

It was his Honour’s view that the transportation of fire burning fuel through the intake shaft in the mine is a foreseeable event constituting a risk.

McMurdo J, stated:

> The fact that the applicant’s interpretation of s 296 could require in an individual case, more than is necessary to achieve an acceptable level of risk does not necessarily demonstrate that the interpretation is incorrect. Rather, the fact that the regulation requires more than is reasonably necessary in an individual case could reflect a preference for certainty and for the avoidance of dangerous conditions from an erroneous judgment by the mine operator about whether the mine does represent an acceptable risk (at 48).

Conclusions of the Court

It was the conclusion of the court that a fire in the intake shaft was a reasonably foreseeable event. Such an event could contaminate the airways throughout the mine and its exhaust airway, at least to the extent or requiring the protective equipment to be worn at all times by a person who was escaping until that person was safe at the surface. This in turn according to the Court, had the potential to substantially, rather than negligibly, affect the utility of the escapeway.

The court concluded that the two entrances at the mine were not separated as required by Section 296(1) of the Regulation and as a consequence, the ultimate result on this occasion was that a third shaft was constructed at a significant cost to the company.

CONCLUSIONS

So what are the conclusions that are to be drawn from all of this? In the case of Greyley what can be absorbed to the extent that one can learn from the decision at the present time, is that the responsibilities and obligations under the law are far reaching, overlapping and interconnected. The very notion of derivative charges that takes place as a flow on from an earlier act or omission shows the unrelenting way that the responsibilities and liabilities under the law can be determined and prosecutions pursued.

Greyley also serves as a timely way for all those charged with statutory obligations to consider the significance of what they do and their own personal exposures under the law. The Grasstrees decision on the other hand, shows that the safety and health law can be open to much debate as it is interpreted for implementation in the workplace.

What the legislation is based on is an environment of social partnership between government, industry and unions to forge common goals within a framework that recognises the pre-eminence given to safety and health, while maintaining economic incentives. An outcome based approach to the legislation assumes that the performance of the parties will take place in a cooperative fashion.

Perhaps in the scheme of things, to have the courts interpret the legislation, as a point of last resort is still the better approach than a system of overly prescriptive regulation. That may be the case, providing the lessons from the decisions of the court are built back into the understanding of the parties and that the intractability of the parties may be softened against an environment that shows time and time again, the benefits of consultation and cooperation, rather than conflict and chaos.

REFERENCES

Coal Mines Regulation Act (NSW), 1982.
Coal Mining Safety and Health Act (Qld), 1999.
Coal Mining Safety and Health Regulation (Qld), 2001.
Construction, Forestry, Mining and Energy Union v Oaky Creek Coal P/L, Supreme Court of Queensland, 26 February 2003.
WorkCover Authority of New South Wales (Inspector Byer) v Cleary Bros (Bombo) Pty Ltd, 110 IR 182.

END NOTE FOLLOWING DECISION

11 MARCH 2005

On 11 March 2005, Staunton J handed down his decision in relation to the penalties in this matter.

Newcastle Wallsend Coal Company and Oakbridge together were fined $1.46 million, that was moderated downwards based in accordance with sentencing law.

The mine manager at the time of the accident and the two other personal defendants were fined $42 000 and $30 000 each respectively.

In reaching its decision the court considered the scope of the defendant’s ongoing obligations and the positive cooperation that has taken place on behalf of the companies.
Wear of Dragline Wire Ropes

D Dayawansa¹, M Kuruppu², F Mashiri¹ and H Bartosiewicz¹

INTRODUCTION

There are three categories of dragline ropes namely, drag and hoist ropes, IBS and main suspension ropes, and dump ropes, which differ mainly by their functions. Previous experience on dragline rope maintenance suggest that some of the problems experienced are common to all three categories of ropes while others are more specific to one or more categories.

Common issues

All three categories of ropes are subjected to fatigue loading, corrosion, and abrasive wear in the drums, sheaves and on the floor in case of dump ropes. The average life of all three categories of ropes has reduced over the last decade although, it is very difficult to pin point the factors that have made the most significant contributions to this life reduction. Recent studies conducted on failed rope samples by Maintenance Technology Institute (MTI) highlighted that rope material and quality issues have a significant effect on rope life. The quality of the wires, including material quality and drawing process, is a critical factor that determines the fatigue performance of ropes. These factors may be related to some of the reduction in rope life that has been experienced during recent times. Another key observation made is the significant variability of life between different mines and machines. The possible factors contributing to these differences are operating conditions/characteristics, maintenance practices and equipment design.

Drag and hoist ropes

The records of hoist and drag rope lives at BHP Billiton Mitsubishi Alliance (BMA) for the two common models of draglines, the BE1370 and M8050, over a period of 20 years showed a service life spread between five to 35 weeks (average 17 weeks) for M8050 hoist ropes, ten to 55 weeks (average 32 weeks) for BE1370 hoist ropes, and two to 30 weeks (average ten weeks) for drag ropes. Around 90 per cent of ropes had life longer than one half of the average life. A very significant difference in the hoist rope life performance between the two models of draglines has also been observed. In addition to rope design and manufacturing issues, there are many other factors that can affect the service life of the drag and hoist ropes. These include:

- design of the rope suspension system (ie sheave and drum groove geometry, sheave and drum diameter, spacing between sheaves, boom point swivel torsion bars);
- maintenance practices (ie lubrication, sheave and drum groove shape);
- rope termination and end effects (ie bending and wear at sheaves, sockets and drum);
- ground conditions (mainly applicable to drag ropes);
- mode of operations (chopping, suspended loads, bucket lift and swing angle); and
- rope retirement criteria (including end-to-end and replacement).

In spite of the past investigations of the effect of rope design on rope service life, satisfactory explanations to the problems experienced by the dragline ropes have not been found. The main reason for this has been the limited scope of these studies, which tended to focus only on a few selected factors affecting rope service life under idealised general conditions rather than investigating the rope performance in an actual application. The problem was further hampered by lack of data on the scatter of rope service life and its correlation to the operating and maintenance practices at the mine sites.

IBS and suspension ropes

The IBS ropes in Marion 8050 machines have had several maintenance issues in the past and, in several instances, they have failed during service. Although the boom is able to support itself temporarily without an IBS rope, this can lead to serious consequences in combination with other critical structural weaknesses that may exist simultaneously. Furthermore, these failures would result in significant amounts of downtime and repair costs. It is very important that operators ensure neither main suspension ropes nor IBS ropes fail during operations.

Dump ropes

The dump rope life can vary from one day to approximately two weeks. This service life has been achieved both with new dump ropes and re-used hoist ropes. The short life of these ropes results in many rope replacements costing unscheduled downtime. This leads to significant downtime as each rope replacement could take a few hours (depending on the maintenance crew mobilisation time). Previous MTI studies have shown that the most significant cause of the short life of dump ropes is due to the natural bucket noddng phenomenon, which leads to high bending and fatigue loading in the dump rope. Unfortunately, this cannot be avoided with the current designs of the bucket jewellery.

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ABSTRACT

Wire ropes are one of the most heavily used components in a dragline. They are subjected to harsh conditions during the regular usage of a dragline in a mining operation. Hoist ropes are subjected to fatigue due to the cyclic nature of load handling as well as due to rope bending over the sheaves and the drum under load. This leads to wire breaks due to fatigue. Accumulation of a number of wire breaks close to each other can have a detrimental effect on the rope. Furthermore, to allow for the increasing demand for higher load capacity coupled with the inconvenience of having very large ropes, the factor of safety is often compromised, which increases the wear rate. Drag ropes are also subjected to heavy loads. More importantly, they are allowed to drag along the rough mine surface subjecting them to external physical abrasion. This makes the life of drag ropes one of the lowest among those used in a dragline. Suspension and IBS ropes are relatively uniformly loaded during their regular usage although they need to withstand dynamic load cycles as well as bending. Hence they tend to last for a number of years on average. The paper analyses the wear types and their severity of each of these rope applications, and suggests methods to determine rope wear rates and the resulting rope life. The paper further gives suggestions for good operating and maintenance practice that can extend the rope life and help reduce the large expenditure associated with every major rope change in a dragline.
Maintenance issues and cost

Recently conducted review by MTI on dragline maintenance costs shows wire ropes (including suspension, IBS, drag, hoist and dump) to be a major driver of maintenance costs. It accounts for ten to 15 per cent of the total maintenance costs and amounts to about $300 000 per annum per machine.

The frequency of major shutdowns for the draglines in the Australian fleet is generally around every five years. Therefore it is preferable to change the main suspension ropes in multiples of five years (say every ten years), covering the duration of one or two shutdown periods. However, the currently used suspension ropes have a wide scatter in life varying from four to eight years (Marion’s closer to four years and BE’s closer to eight years), and hence require a change over in-between shutdowns. It is interesting to note that lower suspension ropes of BE machines (Marion’s only have ropes for upper suspension) can last ten years in most instances but the exact reason for this has not been documented. The cost of lowering the boom alone to replace the suspension ropes, excluding the cost of downtime of ~4 - 5 days, is around $100 000. For most machines, the suspension ropes are generally changed every five years during shutdowns, irrespective of remaining life. The owners/operators are exploring the possibility of selecting suspension ropes that can ideally cover two shutdown periods.

Scope of study

This study focuses mainly on two types of draglines, the BE draglines and the Marion draglines. The accessories that assist the operation of the hoist ropes are the boom point sheaves, the upper and lower deflection sheaves, the mast deflection sheaves and the hoist drum as shown in Figure 1 (a, b and c). For the drag ropes, the accessories that assist in the operation of these ropes include: lower and upper vertical fairlead sheaves, the horizontal fairlead sheaves and the drag drum. Figure 2 (a and b) shows some of the accessories that assist in the operation of the drag ropes. Marion draglines also have most of these components arranged in a slightly different configuration.

HISTORY ANALYSIS

Hoist ropes

In draglines, hoist ropes are used to manoeuvre the bucket during the digging and filling process and to assist in lifting the bucket from the point of excavation to the point of dumping. In so doing, the hoist rope length has to be continuously adjusted to suit the locations of digging and dumping. The adjustment of the hoist rope length, which is primarily held at the hoist drum, also means that the hoist ropes have to constantly pass through deflection sheaves or gantry sheaves as well as the boom point sheaves. Bending over the sheaves occurs at locations such as the boom point sheaves and the gantry sheaves, where the rope has to change its direction. Secondary bending may also occur over the deflection sheaves due to the self weight of the wire ropes, the load carried by the bucket, and the distance between the deflection sheaves. It is mainly the bending over the boom point sheaves that contributes to the fatigue crack initiation and growth that ultimately leads to the fracture of the individual wires in strands of a wire rope. It is the breaking of these wires that finally leads to the discard of wire ropes.

Fig 1 - BE dragline accessories for hoist ropes. (A) Boom point sheave, (B) boom deflection tower and sheaves, (C) mast deflection sheaves.
Marion M8050 data

The average and best hoist rope service lives are obtained by analysing the life of five ropes used on a specific machine prior to a given date. This means that average and best service lives will vary over an extended period of time. Figure 3 shows the moving average hoist rope life for a Marion dragline and demonstrates the variability of hoist rope life over time.

The Marion draglines with a recommended suspended load (RSL) of 130.7 tonnes and 131.5 tonnes use hoist ropes with a diameter of 83 mm, whereas those with an RSL of 137 tonnes use hoist ropes with a diameter of 85 mm. The Marion draglines with an RSL of 137 tonnes also have a larger sized hoist rope drum with a diameter of 2.54 m compared to Marion draglines with an RSL of 131.7 tonnes, which have hoist rope drums that are 2.438 m in diameter. All the draglines currently use six-strand ropes, usually of tensile grade 1770 MPa. The hoist rope drum to rope diameter ratios are 29.4 and 30 for the machines with rope diameters of 83 mm and 85 mm, respectively.

BE 1370 data

Figure 4 shows the average hoist rope service life for a BE 1370 dragline over a long period of time and also demonstrates the variability of hoist rope life over time. The BE draglines have RSLs ranging from 93 tonnes to 163.3 tonnes and use rope sizes ranging from 73 mm to 92 mm in diameter. Six strand wire ropes are also used with steel grades of 1570 MPa and 1770 MPa. The hoist drum to hoist rope diameter ratios in BE draglines ranges from 30.4 to 35.5.

Comparison of hoist rope data

The average hoist rope data for Marion and BE draglines is shown as a scatter in Figure 5. The data shows that BE machines, on average, have a better hoist rope service life as compared to Marion 8050 draglines. Although variability of hoist rope service life is to be expected over a period of time, there is a clear trend in this data. A mean hoist rope service life is about 6.5 million bench cubic metres (BCM) for BE draglines as compared with a mean hoist rope service life of about 4.2 million cubic metres for Marion draglines. The mean service life for hoist ropes in BE draglines in the sample is about 1.5 times the mean service life for hoist ropes in Marion draglines.

Factors influencing service life of hoist rope data

Sheave diameter to rope diameter ratio

One of the factors that influence the bending fatigue strength of hoist ropes bending over sheaves is the sheave diameter to rope diameter ratio. A plot of the average hoist rope service life versus the boom point sheave diameter ($D_{bps}$) to rope diameter ($d_r$) ratio is shown in Figure 6. The Figure 6 shows that for the majority of the data a higher $D_{bps}/d_r$ ratio results in better service life for the hoist ropes. However, some of the data shows that although their $D_{bps}/d_r$ ratios are high, there is no corresponding benefit in the service life. This shows that despite the favourable $D_{bps}/d_r$ ratios, other factors can also negatively influence the service life of hoist ropes bending over sheaves in hoist rope systems. The scatter of average hoist rope service life for machines with the same $D_{bps}/d_r$ ratios also confirms that other factors influence the hoist rope service life.
Bending at gantry sheaves in Marion draglines

According to the rope configuration, additional bending of the hoist ropes occurs at the gantry sheaves in Marion draglines. This is likely to negatively impact on hoist rope service life resulting in a lower hoist rope service life in Marions compared to BE draglines.

Fleet angle between gantry sheaves and drum

The fleet angle between the gantry sheaves and the hoist drum in Marion draglines also causes wear to occur at the bottom quadrants of hoist ropes. This phenomenon has been observed during rope inspections.

Drag ropes

Drag ropes work in conjunction with hoist ropes to manoeuvre the bucket during the digging process to fill the bucket, during the hoisting of the bucket to the location for dumping as well as during the dumping process itself. In both Marion and BE draglines, the drag ropes bend over the drag drum and the fairlead sheaves and are connected to the bucket through wedge sockets. The positioning of the sheaves and the rope configuration is different in each type of dragline. Although bending over the fairlead sheaves may influence the service life of ropes, it is the wear that occurs due to the drag ropes passing through spoil that is likely to be a major factor in the service life of drag ropes.
Comparison of drag rope data

The average drag rope service life for Marion draglines is illustrated in Figure 7. The data analysis shows that the mean service life for Marion draglines is slightly higher than the mean service life for BE draglines. A mean service life of 2.7 million cubic metres was obtained for the Marion draglines compared to 2.3 million cubic metres for BE draglines. However, the scatter of the average drag rope service life (Figure 7) shows that except for those average rope service lives that define the upper and lower bound of this data, most of the service lives for the BE and Marion draglines fall within the scatter band between two million and three million cubic metres.

Factors influencing service life of drag ropes

Condition of blasted overburden

The average service life of drag ropes depend on the condition of the blasted overburden. During digging operations, the drag ropes can run through the spoil collecting soil or rock particles on its lubricating surfaces. The soil and rock particles can become wedged between individual wires in a strand or between individual strands. The load in the drag ropes can cause the abrasive soil and rock particles to be compressed between two solid surfaces of the individual wires or strands. The high contact pressure produces indentations and scratching of the wearing...
surfaces and fractures and pulverises the abrasive ore particles (Hawk and Wilson, 2001). This type of abrasion is classified as high stress or grinding abrasion. The larger rocks that result from improper blasting can result in the wear of drag ropes as well as cutting and tearing types of wear, in which small chips of metal are removed from the wearing surface by the movement of the sharp points of rock, under considerable pressure, over the wearing surface (Hawk and Wilson, 2001). This type of abrasion is called gouging abrasion. Wear of drag ropes therefore depends on the characteristics of the blasted overburden.

Fairlead sheave diameter to drag rope diameter ratio

It is well known that the service life of a rope bending over a sheave depends on the sheave diameter to rope diameter ratio. The average hoist rope service life shows that service life increases as the D/d ratio becomes large. Figure 8 shows that for drag ropes, a higher D/d ratio does not result in an improved service life. Therefore the dominant mode of failure in those ropes is a result of the abrasion that occurs due to the interaction of the rope and overburden during digging operations.

**CAUSES OF ROPE DISCARD**

Records from OneSteel Wire Rope (2003) show the reasons for discard of some of the wire ropes that are used as either hoist or drag ropes. The major discard indicators classified by OneSteel Wire Rope (2003) are as follows:

1. mechanical damage,
2. damage at sheaves,
3. damage at sockets,
4. removed for operational reason,
5. damage at drum, and
6. influence of equipment operation on wire rope life.
About 26 per cent and 40 per cent of the recorded discard reason for hoist ropes was due to damage occurring at the sheaves in Marion and BE draglines, respectively. A larger percentage of the discard reason for hoist ropes was also due to damage occurring at the sockets as well as mechanical damage. Other reasons for discard such as operator error, worked to destruction, removal for operational reasons, warranty claim and damage at the drums are not very common for discard of hoist ropes.

About 58 per cent and 48 per cent of the recorded discard reason for drag ropes was attributed to damage at the sockets for Marion and BE draglines, respectively. A significant proportion of drag ropes was also discarded due to operator error and mechanical damage in BE draglines. Other than mechanical damage, drag ropes discarded because of operational reasons were also significant in Marion draglines.

In the case of suspension and IBS ropes, the major reason for discard is the wire breaks due to fatigue. In particular, the end sockets are critical points where large numbers of wires tend to fail. However, the failure of ropes is not as rapid as in the case of hoist and drag ropes. The wire breaks usually can be monitored if an appropriate non-destructive method is employed.

**FACTORS AFFECTING ROPE LIFE**

The key parameters affecting dragline wire rope service life are:

- wire rope loading,
- wire rope construction and mechanical properties,
- sheave and drum design and configuration,
- maintenance of wire ropes and equipment in contact with wire ropes,
- storage and handling of ropes, and
- operator practice.

This section will broadly review each of these parameters in order to establish the reasons for significantly lower service life of hoist wire ropes on Marion 8050 draglines compared to BE 1370 dragline wire ropes.

**Wire rope loading**

Past analysis of the static rope forces under a range of the hoist and drag rope pay-out lengths using PCDRAG software (Sour and Shanks, 1995) have shown hoist rope tensions in excess of 200 tonnes (for average bucket payload), with the bucket positioned well away from the tightline situation, and in excess of 300 tonnes, when the bucket is positioned near the tightline conditions. Thus, for an 83 mm diameter hoist rope, the average loads per nominal rope metallic area can be in the order of 49.4 kg/mm² to 74 kg/mm². It is expected that dynamic loading on dragline ropes are likely to be significantly greater and will be governed by digging conditions and operational practices. Typical maximum dragline rope loadings in terms of specific stress and factor safety are summarised in Table 1 and it shows that the actual static and dynamic loading far exceeds the allowable factor of safety.

**Wire rope construction**

The ropes used for both hoist and drag are 83 mm in diameter. The recommended (BHP ropes) wire rope construction for hoist ropes is 6 × 25 FW (12/6 + 6F/1) and 6 × 49SF (16/16/8 + 8F/1) for drag ropes. BHP ropes have claimed to have trailed some of their own eight stranded products and monitored the performance of the eight stranded ropes supplied by other manufacturers. These tests, apparently, have shown no evidence of any improvement in wire rope life or that they impart any advantage to the operation or maintenance of equipment. However, other studies have shown that the use of eight-strand ropes increases the life of hoist ropes (Golosinski, 1994).

**TABLE 1**

Typical maximum dragline rope loadings.

<table>
<thead>
<tr>
<th>Rope type</th>
<th>Factor of safety</th>
<th>Approximate stress (kg/mm²)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drag ropes</td>
<td>5:1</td>
<td>36</td>
</tr>
<tr>
<td>Hoist ropes</td>
<td>8:1</td>
<td>22</td>
</tr>
</tbody>
</table>

The usual tensile grade for mining ropes is 1770 MPa. However, hoist ropes will generally have the outer wires with a lower grade than the nominal tensile grade for the rope, whilst higher tensile grade wires will be used in the drag rope in order to improve the wear resistance. Note that the fatigue endurance is greatest around 1600 to 1700 MPa. Hence, an increase in drag rope wire tensile strength must be carefully monitored due to the potential of increased fatigue type problems at the sockets. It is well known that ropes having high tensile strength are subjected to increased fatigue (Evans and Chaplin, 1997).

**Sheave drum D/d ratio**

Sheave and drum size can have a significant impact on the rope bending fatigue and its service life. In general, the rope life will improve as sheave and drum sizes are increased. The rope manufacturers recommend that the sheave and drum sizes are maintained in the range of 25 to 30 rope diameters. Reducing the size of drum and sheaves below 25 rope diameters is claimed to have a profound effect on rope life (OneSteel, 2001).

**Maintenance of ropes and equipment in contact with the ropes**

The key parameters affecting rope service life from the maintenance point of view are:

- rope condition inspection,
- sheaves/drum conditions inspections,
- rope and sheave lubrication practices and inspection,
- rope end for ending practices, and
- re-socketing practices.

**Rope condition inspections**

Review of dragline inspection schedules has shown that the rope/sheave inspection practices vary significantly between the mine sites. Visual examination of ropes by the dragline operators is generally carried out. The objective is to discover unusual damage caused by some accident, such as broken or damaged wires, a kinked rope or loose rope fittings. End connections and fittings are also checked. Assessment of rope condition requires visual examination of the entire length of rope. This includes cleaning the rope at marked points, noting rope diameter, external corrosion, broken wires and strand slackness. The point of connection of the rope to the drum is also examined carefully for broken wires and slackening of bolts or clamps. The bucket/socket connections are also examined. Where possible, non-destructive tests must be used to determine internal wire breaks.

**Sheave/drum condition inspection**

The key parameters to be examined are correct groove size and profile, and alignment. Shaeves that are badly worn can pinch a new rope or produce undercut shoulder that will rub against the rope and severely impact on rope life. Uneven wear of the sheave flanges will often indicate an alignment problem. Heavy plastic wear early in the rope’s life on the section running over sheaves is also an indication of either undersize sheave grooves, misaligned sheaves or sheaves not rotating freely. Continual use
of undersize sheaves will eventually lead to broken wires due to fatigue. These inspections are recommended to be carried out annually or more frequently if rope wear is evident.

**Lubrication practices and lubrication system performance inspection**

Thorough lubrication of dragline ropes is of great importance in order to achieve good rope service life. According to BHP wire ropes experience, the rope life can be reduced from 40 per cent to 50 per cent if the ropes are not lubricated beyond what was applied during ropemaking. Inadequate lubrication could also seriously impact on the wear of the sheaves and drums. The type of lubricant used and means of lubricant application can also have a significant impact on the rope life.

**Rope end-for-ending practices**

Most dragline users swap the two ends of a rope about half way through the rope life. The purpose of rope end-for-ending practice is to maximise service life of ropes. The practice recognises the fact that rope wear and its deterioration is not symmetric along its length. The timing for end-for-ending is crucial towards it being successful practice. It was found through experience that the best time for end-for-ending a set of ropes is, in the case of hoist ropes, before any fatigued broken wires occur and, in the case of drag ropes, before a failure of any strand.

**Re-socketing**

The need for re-socketing arises from the fatigue and wear damage at the socket entry. Drag ropes in particular are prone to this mode of deterioration and are greatly affected by the digging operations occurring at the time. Re-socketing should be carried out before the number of broken wires reaches the rope discard criteria. Regular inspections of rope socket connections are thus required to minimise the possibility of either a strand failure or rope failure at these points. An anchor crop close to the socket can be done if the damaged area is not extensive and this will save on the amount of rope discarded and may give an extra crop.

**Storage and handling of ropes**

Storage and handling of ropes so as to avoid any damage to the wires and strands is very important. Also the application of lubricants prior to storage will minimise the possible corrosion.

**Influence of equipment operation on wire rope life**

The operator of any equipment has a role to play in minimising unwanted maintenance problems that can arise due to the operating practice. This is especially true for mining equipment, considering the harsh environment where they are put to use.

Some of the essential points are (OneSteel, 2001):

- Dragging the ropes through spoil piles must be minimised.
- Maintain on line digging wherever possible by aligning the bucket, boom point and the fairleads. Avoid slewing during digging.
- Minimise chopping to avoid rope friction with bucket chains.
- Bucket movements that can cause impact or dynamic loads on the ropes should be avoided.
- Worn, broken or missing bucket teeth will impact on rope loading and should be replaced.
- Do not let the ropes rub against the rock, slope or any foreign body.
- Do not let the rope slack – can cause winding problems on the drum.
- Many machines are fitted with ‘tight line’ limits based on permissible rope loads. The practice of operating away from this situation will reduce the load on the rope and likely increase the operations of the dragline overall.
- Good blasting practice, selecting manageable distance and depth of material removal – they all help minimise the duty on the machine.

**CONCLUSION**

Proper selection, installation and maintenance of ropes and the related parts is essential for maximising ropes service lives and to improve the reliability of draglines. Maintenance of the rope systems involves condition monitoring as well as preventive actions, such as lubrication, end-for-ending and re-socketing, whenever they are required. Furthermore, operator care impacts on the reliable operation of the draglines and improves the life of ropes.

**ACKNOWLEDGEMENTS**

This paper is a brief report of an extensive work, which is currently in progress. Supporters for the work include the Australian Coal Association Research Program (ACARP), Monash University, mines using draglines, in particular, the BHP Billiton Mitsubishi Alliance (BMA), the manufacturers of wire rope, in particular OneSteel Wire Rope, and the manufacturers of draglines, in particular, BE International.

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‘Keep the Cream’ — Reconciling Coal Recovery at BMA Goonyella Riverside
S N Johnston1 and M J Kelleher2

ABSTRACT
CREAM is an acronym for coal recovery, evaluation, analysis and management and is a business improvement project at BMA’s Goonyella Riverside mine that is focussed on maximising coal recovered.

In August 2003, Goonyella Riverside was set a challenge to quantify coal loss and dilution and their relationship to cost, revenue and ultimately profit. While the mine was confident in its site forecasts of recovery (and loss), it was clear that the traditional measurement and analysis systems were inadequate to firstly, identify key loss areas and mechanisms and secondly, successfully quantify the benefit of various coal recovery initiatives.

The path that BMA Goonyella Riverside has travelled to maximise coal recovered can be divided into four key steps:

1. Mapping the process – This step involved the development of a coal flow process map for Goonyella Riverside that stepped through each component of the process from the coal reserve to the customer and identified key measurement points for the purposes of reconciliation.

2. Understanding coal loss and identifying opportunities – This step involved conducting two detailed Reserve to Customer projects at the mine site to track a block of coal from the reserve to the customer and to gain a greater understanding of the relationship of dilution, coal loss mechanisms, revenue and costs associated with recovering additional coal.

3. Coal data systems development – Two important coal data systems have been developed and implemented at Goonyella Riverside in the past twelve months, namely Snowden’s Coal Reconcilor and SeamFix. These systems enable coal losses and dilution to be quantified and reconciled back to the initial coal reserve.

4. Implementation of loss reduction initiatives – This step involved the formation of a business improvement project referred to as ‘CREAM’ in August 2004, for the purposes of improving pit coal recovery and maximising business value. To date a series of coal recovery trials have been run in a number of pits at Goonyella Riverside that have yielded significant improvements in business value.

The key to the success of this project has been the discipline to follow a defined process map, management commitment through resourcing and shared key performance indicators, a successful acceptance strategy resulting in holistic ownership and the implementation of sustainable reconciliation systems. Coal loss and dilution are now quantifiable and their relationship to cost, revenue and ultimately profit is well understood. This paper discusses the process that Goonyella Riverside has adopted to maximise coal recovered.

BACKGROUND
Goonyella Riverside is the largest coking coal producer of nine mines operated by BHP Billiton Mitsubishi Alliance (BMA) in the Bowen Basin. Goonyella Riverside is located 190 km south-west of the Hay Point port facilities and 30 km north of the Moranbah township.

The economic coal seams within the Goonyella Riverside lease area are contained within the Late Permian Moranbah Coal Measures. Three seams are typically mined: the Goonyella Upper, Middle and Lower Seams. All are high quality, medium volatile coking coals and are widely recognised for their superior coking characteristics. The combined Goonyella Riverside leases have open cut reserves in excess of 600 million tonnes. The Goonyella lease has an in situ resource of approximately 1.6 billion tonnes.

BMA Goonyella Riverside currently operates a stripping fleet of four electric rope shovels (1 × P&H 2800, 1 × P&H 4100A, 2 × P&H 4100XPBs) and seven draglines (2 × BE1370s, 2 × BE1350s, 2 × Marion 8050-47s and 1 × Marion 8050-12). A simple schematic of the mining operation can be seen in Figure 1.

Goonyella Riverside has the capacity to produce over 13 million tonnes of coking coal product. To achieve this production, 180 million cubic metres of overburden will be moved (including rehandle) to uncover around 18.5 million tonnes of raw coal. It is clear that given the existing strip ratio that any incremental coal recovered from this process will add significant value to the business (a one per cent coal recovery improvement at Goonyella Riverside, equates to 185 000 tonnes of raw coal).

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FIG 1 - Goonyella Riverside Mine stripping operations.
THE PATH TO MAXIMUM COAL RECOVERY

Coal damage, loss and dilution are inevitable outcomes of the mining process when utilising large-scale mining equipment. The path that BMA Goonyella Riverside has followed to maximise coal recovered can be divided into four key steps:

1. mapping the process,
2. understanding coal loss and identifying opportunities,
3. coal data systems development, and
4. implementation of loss reduction initiatives.

These four steps highlight Goonyella Riverside’s journey through the process of reconciliation and business improvement to ultimately maximise business and shareholder value. Each of these four steps is discussed in greater detail below.

MAPPING THE PROCESS

Development of the BTRAK guide

The Business Improvement and Optimisation group within BMA facilitated a joint corporate/mine site project to map the coal flow process from the reserves to the customer, identifying key measurement points for the purposes of reconciliation. This was later developed into the BMA guidelines for tracking coal referred to as ‘BTRAK’. A gap analysis was carried out on the existing measurements and analysis processes and an action plan was created to rectify shortfalls.

UNDERSTANDING COAL LOSS AND IDENTIFYING OPPORTUNITIES

The mine site undertook two Reserve to Customer projects to track a block of coal from the reserve to the customer and to gain a greater understanding of the relationship of dilution, coal loss, revenue and costs associated with recovering additional coal. Direct costs associated with recovering the coal and processing additional dilution as well as indirect costs associated with scheduling constraints and inefficiencies generated by these recovery processes were analysed. By understanding the mechanisms through which coal is lost, continuous improvement initiatives may be developed on-site to maximise coal recovered.

After the completion of the two Reserve to Customer trials, Goonyella Riverside reviewed its site coal assumptions used for forecasting and reserving.

Reserve to Customer (R2C) projects

Reserve to Customer trial 1 – Ramp 13 North Strip 22 Goonyella Lower Seam

In August 2003, the first Reserve to Customer (R2C) project was performed in Ramp 13 North Strip 22 Goonyella Lower Seam to define and demonstrate the engineering and operational requirements to monitor and manage the coal production chain to maximise shareholder value (Reserve to Customer Project, 2004; 4; Scott et al., 2004).

The objectives of the mining recovery trial were as follows (Reserve to Customer Project, 2004: 11; Scott et al., 2004):

- to compare the quantity and quality of a target block of coal with the parameters defined from exploration;
- to track the coal mined from the target area through the preparation plant to the product coal stockpile;
- to identify the locations and mechanisms of loss and dilution;
- to quantify mining recovery, breaker performance and preparation plant yield;
- to compare the overall recovery of saleable coal with that expected from the reserves;
- to recommend any changes to the current planning, measurement, mining, handling or washing practices that would improve future performance and finally; and
- to identify the resources required for similar analyses to be performed reliably and routinely.

After the coal had been uncovered in the trial area, thorough surveys were performed to accurately quantify the coal lost during the dragline recovery and coal mining phases. During this exercise, cores were taken from 25 in-pit drill holes. The cores did not contain coal lost from the top of the seam in the overburden removal process and thus the full seam data was recreated using other core information. The quantity and quality of the in situ coal was then compared to the reserve model where it was found that the two estimates of the in situ tonnage agreed within one percent.

From the Ramp 13 North Strip 22 Goonyella Lower Seam trial it was found that the total loss could be distributed between four key areas: low wall coal wedge, top of coal edge, top of seam and floor of seam. Table 1 outlines the distribution of total coal loss from this trial.

<table>
<thead>
<tr>
<th>Loss area</th>
<th>Percentage of total loss</th>
</tr>
</thead>
<tbody>
<tr>
<td>Low wall coal wedge</td>
<td>27.42%</td>
</tr>
<tr>
<td>Top of coal edge</td>
<td>6.45%</td>
</tr>
<tr>
<td>Top of seam</td>
<td>48.39%</td>
</tr>
<tr>
<td>Floor of seam</td>
<td>17.74%</td>
</tr>
</tbody>
</table>

It should be noted that only 45 per cent of total loss takes place in the mining process with the remainder lost during the overburden removal processes. The total actual losses were found to be consistent with forecast actual loss; however, other key coal assumptions required further analysis.

Figures 2, 3 and 4, highlight the various types of coal losses that were analysed and measured during the Ramp 13 North Strip 22 Goonyella Lower Seam trial.

Reserve to Customer trial 2 – Ramp 10 North Strip 24 Goonyella Middle Seam

In February 2004, the second Reserve to Customer trial was performed in Ramp 10 North Strip 24 Goonyella Lower Seam for the purpose of confirming the key findings deduced from the first Reserve to Customer trial, to demonstrate that coal can be recovered using coal recovery initiatives and to further develop pit loss measurement techniques.

Similar to the first Reserve to Customer trial, extensive survey and geological control were crucial in successfully quantifying coal lost from the top of coal, low wall edge and floor as well as dilution due to the various mining processes from drill and blast through to eventual coal mining.

Due to an increased focus on coal recovery during the second Reserve to Customer project, it was apparent that there was significantly less damage to the top of coal and the low wall edge in comparison to the first Reserve to Customer project. In this trial, an excavator was utilised to expose the low wall edge of coal (an excavator dug low wall trench), which provided extremely favourable results and warranted further assessment. An important finding from this trial process was that the cost of utilising an excavator to dig the low wall trench is extremely cheap in comparison to the economic value obtained from the additional coal recovered.
Conclusions from the Reserve to Customer trials

The key findings and results from the two Reserve to Customer trials were as follows:

1. That coal can be tracked on a block-by-block basis from Reserve to Customer.
2. That the existing data and information gathering systems were under-utilised.
3. That a change in loss calculation philosophy was required. This involved a move from the traditional calculation (reserve minus mined void equals loss) to the direct analysis of loss itself.
4. That additional coal can be recovered at minimal cost by the implementation of coal recovery initiatives.
5. The in situ reserves and forecast saleable coal were within one per cent of actual.
6. For the purposes of reconciliation, coal data systems development was required to reduce engineering and surveying requirements.
7. A review of the Goonyella Riverside key coal assumptions for forecasting and reserving was required.
8. The development of a sustainable and an ongoing pit recovery project would be required to implement coal recovery initiatives to maximise business value.

Review of site coal assumptions

Based on the key findings from the two Reserve to Customer projects, the business improvement and optimisation group at Goonyella Riverside facilitated a review of the site coal assumptions for forecasting and reserving. The inconsistencies between short-term and long-term forecasting were resolved and the findings were implemented immediately.
COAL DATA SYSTEMS DEVELOPMENT

An integrated coal data management system is essential to efficiently and effectively reconcile coal processes. During the past twelve months, two important coal data systems have been developed and implemented at Goonyella Riverside they were Snowden’s Coal Reconcilor and SeamFix. The key goal of the implementation of these systems was to allow sustainable reconciliation with a minimal increase in engineering and surveying resources.

Snowden’s Coal Reconcilor

In April 2004, Goonyella Riverside commenced a trial using Snowden’s Coal Reconcilor for the purposes of integrating all site coal data to enable tracking and reconciliation of coal from the reserve model to the product stockpile.

Snowden’s Coal Reconcilor is an Internet based software program that directly imports coal mining haulage data from the on-site fleet management system (FMS), reserving information from the XPAC mine model and preparation plant data from the mine information system. It allows input from survey, coal, dragline and prestrip planners on updated coal parameters and further input from coal geologists. The Internet based format of Snowden’s Coal Reconcilor is extremely user friendly and allows the identification of coal losses and targeting of continuous improvement initiatives to maximise coal recovered. The most significant hurdle in the implementation of this software was the modifications required to existing coal data systems and the roll out of consistent nomenclature. This trial will be concluded in March 2005.

SeamFix

During the two Reserve to Customer trials, the surveyors at Goonyella Riverside were required to perform an unsustainable level of daily surveys due to the level of accurate measurement required for the purposes of reconciliation. It was concluded from the Reserve to Customer projects, that the traditional measurement of the mined void for the purposes of reconciliation added little value as the error in the mined void calculation was often found to be greater than the loss to be reported. In response to the development of a new loss measurement philosophy, the chief surveyor at Goonyella Riverside (Damien Vella) designed a graphics based survey software tool called ‘SeamFix’.

SeamFix exploits the very regular and consistent seam structure along the strike as measured in exposed high-walls to interpolate the strike undulation string between surveyed high-walls for any position on the dip. This string is then indexed up or down to visually fit the actual surveyed roof and floor that reveals structural undulations along the dip. This indexing is at the discretion of the experienced pit surveyor who has made the measurements and is familiar with the existing geology and pit conditions.

The structural roof grade model is reconstructed using the available survey data and observations of pit conditions and photographs. The top of coal roof survey is used to vertically index the interpolated strike lines where the seam is not damaged. An interactive graphical interface provides the means to make these adjustments to the grade model with a high level of confidence by providing an instantaneous visualisation in both profile and section views at any place in the pit. Shunted coal blocks, residual wedges and other abnormal features are measured by digitising in section view on slices taken at suitable distance increments along the pit. Surface comparison and volumetrics are performed on a one by one metre grid with output in standard CSV and Excel formats.

SeamFix is a computational component in a heuristic surveying methodology developed to meet reconciliation requirements. This program provides the means of rebuilding the most probable structural roof using the natural undulation along the strike and the surveyed top of coal surface. SeamFix was primarily designed to assist in calculating losses from coal seams where the successful use of the program is dependent on the user’s surveying skills and a thorough knowledge of existing pit conditions. Figures 5 and 6 highlight SeamFix’s ability (in cross-section and long-section) to reconstruct the most probable grade control surfaces based on the surveyed seam structure and the defined block seam thickness.

This software program has allowed the surveyors at Goonyella Riverside to appropriately model top of coal, low wall and low wall shunting, floor and dilution losses as well as reconcile a pit within a relatively short period of time. With the implementation of SeamFix, the surveyors at Goonyella Riverside now have a set process to assist and enable mine-site coal reconciliation.

Fig 4 - Floor of coal loss during the Ramp 13 North Strip 22 Goonyella Lower Seam trial.
Improving pit coal recovery and maximising business value.

In August 2004, a CREAM project team was formulated for the purposes of measuring and managing coal reconciliation, process development and technical services departments (that is, drill and blast, recovered and requires that all functions of the mining operations and technical services personnel from operators through to managers. The team and the team charter were developed, the aims of the CREAM project were derived as follows: to maximise business value.

To assure alignment and direction during the regular Project CREAM meetings, the team developed a process roadmap through which the project would be taken from the beginning through to project completion.

The steps involved in the CREAM process map were as follows:

1. The development of a diverse team – the CREAM team consisted of both mine operations and technical services personnel from operators through to managers.
2. The development of a Project CREAM charter – the team charter highlights the problem statement, project goals, definition and milestones, the team members and project sponsors.
3. A root cause analysis – this process allows the team to identify the root causes of problems that exist within the mining process.
4. Solutions generation phase – this process involved the brainstorming and development of coal recovery initiatives based directly from the findings deduced from the root cause analysis.
5. A payoff matrix – by plotting the solutions generated graphically (ease of implementation versus impact on coal recovery), those initiatives that were the easiest to implement and had the greatest impact on coal recovery were initially implemented.
6. A stakeholder analysis – as a team a stakeholder analysis was undertaken to identify the communications target, so as to ensure project success.
7. The development of an acceptance strategy – this phase involved extensive communication to mining operations and technical services personnel so as to obtain buy-in and support to the project.
8. Presentations of CREAM coal recovery initiatives and findings to mining operations and technical services personnel.
10. Reconciliation – this process involves the reconciliation of CREAM coal recovery initiatives utilising coal reconciliation software (Snowden’s Coal Reconcilor and SeamFix) and survey control.
11. Feedback on the results and findings to the mining operations and technical services personnel was made.

**Root cause analysis**

By referencing the conclusions deduced from the Reserve to Customer trials the areas of high coal loss were analysed and the primary mechanisms were identified.

Once the team and the team charter were developed, the aims of the CREAM project were derived as follows: to maximise business value, to change the current mine site culture from a dirt driven culture to a ‘Keep the CREAM’ culture, to accurately measure coal loss (through the use of reconciliation software to minimise the use of surveying and engineering resources) and finally to decrease the amount of contract prestrip on site. The decrease in the amount of on-site contract prestrip was advertised to operators as it is currently visually evident on-site and it was reiterated to them that by mining more coal there is less demand for expensive, future contract prestrip.

**IMPLEMENTATION OF COAL RECOVERY INITIATIVES**

**Project CREAM**

**Introduction**

CREAM is an acronym for coal recovery, evaluation, analysis and management and is an operating excellence project at BMA’s Goonyella Riverside mine that is focussed on maximising coal recovered. In line with BMA’s Health, Safety, Environment and Community (HSEC) objective of ‘Zero Harm’ this goal has been incorporated into the vision of CREAM which is ‘100 per cent coal recovery safely’.

The project is a team-based approach to maximising coal recovered and requires that all functions of the mining operations and technical services departments (that is, drill and blast, prestrip, draglines and coal mining) work together to develop separate coal recovery initiatives to maximise business value.

**Process development**

In August 2004, a CREAM project team was formulated for the purposes of measuring and managing coal reconciliation, improving pit coal recovery and maximising business value.

SeamFix applies the well proven and readily understood calculation method of cross-sectioning. The algorithms that dynamically present sections for viewing on the screen are the same used for volumetrics. What you see is what you get.

Seam Thickness for Block

SeamFix data view in cross-section. (Fig 5)

The structural roof grade model is reconstructed using all available survey data and observations of pit conditions and photographs. The top of coal survey is used to vertically index the interpolated strike-lines where the seam is not damaged. SeamFix provides the means to make these adjustments to the grade model with a high level of confidence by providing an instantaneous visualisation of all structural and surveyed surfaces at any place in the pit.

Seam Fix data view in cross-section. (Fig 6)

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**Coal2005 Conference**

Brisbane, QLD, 26 - 28 April 2005
was conducted on each area. Loss processes were tracked back to their fundamental causes. Through this process, several root causes not previously identified were clearly impacting coal recovered. These included planning processes such as geological model updates for overburden drill planning, ultimately causing test holes to coal and subsequently top of coal damage.

This analysis was vital in the team’s understanding of the different mechanisms by which coal is lost and from this process the team was able to move into the next phase, solutions generation.

**Solutions generation**

During the solutions generation phase, the team undertook brainstorming sessions to determine specific coal recovery initiatives that focussed on the individual findings deduced from the root cause analysis. All ideas and solutions were welcomed and considered during this process where ‘out of the box’ thinking was encouraged. Numerous solutions for lowwall, lowwall trench and top of coal were generated from this process. In all, thirty lowwall and lowwall trench and thirty-five top of coal recovery solutions were formulated.

**Development of payoff matrix**

After the solutions generation phase, a payoff matrix was developed where the ease of implementation of coal recovery initiatives versus their impact on coal recovery was plotted. The team identified those initiatives that were the easiest to implement and had the greatest impact on coal recovery. These initiatives were then prioritised for further analysis. As can be seen in Figure 7, the various solutions generated from the CREAM project were plotted to develop a payoff matrix.

**Acceptance strategy**

The acceptance strategy phase is the most important phase of the entire project. If buy-in from operators and site personnel is not obtained, the project will fail. The team’s acceptance strategy initially began with a series of marketing campaigns to spark operator interest, which involved the development of ‘CREAM?’ posters that were pinned to the noticeboards in the crew start-up areas. Utilising these posters raised many questions and queries from operators about the project. This was a successful means of obtaining initial buy-in and support from operators.

The next phase of the acceptance strategy was tailored presentations to the mining operations and technical services work groups and crews. These presentations set out to answer questions generated by the previous marketing campaign. The presentations were simple and informative and highlighted what each individual operator could do to maximise coal recovered. The adoption of feedback and ideas generated from these meetings was vital in obtaining project acceptance from operators.

During the acceptance strategy phase, ‘pit custodians’ were appointed for the CREAM project. Pit custodians included drill and blast, dragline, contract prestrip, coal mining engineers and supervisors. Pit custodians were asked to discuss coal recovery in their daily contact with operators and work groups along with their usual safety and production messages. Regularly talking about coal with operators highlights the mine’s focus and commitment to its vision of ‘100 per cent coal recovery safely’. This has been an effective tool in changing the culture from that of a dirt driven culture to a ‘Keep the CREAM’ culture.

Management support towards coal reconciliation has been vital in the success of this project to date. Recently, management at Goonyella Riverside developed shared key performance indicators between all stakeholders in mining operations and technical services for coal recovery improvements. This has further aligned all parties to maximise coal recovered.

From the acceptance strategy, it was decided to implement a newsletter on a weekly basis, which would be used as a simple communication tool to mining operations and technical services. The CREAM newsletter highlighted the following: areas around the mine site that required scavenging, feedback on coal damage and loss, feedback from the weekly CREAM meetings, data analysed from the mine-site reconciliation systems, safety messages and alerts to coal mining crews.
Implementation of Project CREAM trials

The initiative implementation phase will be an ongoing process. Short-term and initial gains were significant however fine-tuning and selection of the best initiatives will take considerable time. It was clear that different solutions were suited to different geology, equipment, technique, schedule constraints and conditions. With so many coal recovery initiatives identified, the challenge was to find the best solution for the identified variables. Solutions were targeted based on these variables and a series of trials were scheduled. Recovery performance was analysed using the recently implemented data systems and a cost/benefit analysis undertaken to benchmark these solutions against other coal recovery solutions. Some solutions that were easy to implement in most existing applications were implemented immediately. An example of this was a review of the top of coal cleanup procedure.

CONCLUSION

Over the past two years, Goonyella Riverside has added significant value to its business through the calculated implementation of coal recovery initiatives. From the defined process of reconciliation, the mine is now in a position to benchmark the performance of specific initiatives or combinations of initiatives in differing conditions to maximise net present value. Key to the success of this project has been the discipline to follow the process map, management commitment through resourcing and shared key performance indicators, a successful acceptance strategy resulting in holistic ownership and the implementation of sustainable reconciliation systems. Significant benefits from optimal coal recovery can be returned to the business as profit through either reduced stripping or additional coal production depending on the current market environment. The ‘Keep the CREAM’ culture at Goonyella Riverside gives all employees the satisfaction of contributing to optimal, quantifiable coal recovery.

ACKNOWLEDGEMENTS

BMA Goonyella Riverside is thanked for their permission to publish this paper. Damian Vella, the BMA Reserve to Customer team and Snowden are acknowledged for their continued effort and support. Scott Mine Consulting Services are acknowledged for their technical support in regard to the Reserve to Customer projects.

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Quantification of Opencast Potential Within the Waikato Coalfields Using Pit Optimisation Software

A Prentice

ABSTRACT

Solid Energy New Zealand (SENZ) uses a 20+ year period for integrated planning of all its business activities. It has an ongoing program of coal resource assessment to optimise future mines within this 20 year period, using a six step resource and mine planning process. Desktop review and conceptual study at preliminary Levels 1 and 2 are based on general coal resource information. Further coal resource investigations are carried out for the more detailed evaluations in secondary assessment, feasibility study, feasibility study and detailed engineering from Levels 3 to 6.

At Levels 1 and 2, completing conceptual planning, including preliminary resource evaluation and economic appraisal to the target cost uncertainty level of ±33 per cent, is challenging in New Zealand’s highly variable geological and geographical conditions. Where coal deposit geometries are relatively simple, approximate methods for determining pit limits using overburden strip ratios, seam thickness, physical boundaries, and quality cut-offs may be sufficient. Many New Zealand coalfields are however complicated structurally, with multiple seams and extensive folding and faulting. Coal seams are characterised by variable dip, thickness, and quality over relatively short distances. In the North Island’s Waikato coalfields, coal is typically overlain by weak sediments and clay rich strata that necessitate very flat cut and fill slope angles. The proximity of these deposits to major infrastructure (including towns, highways and rail lines), significant environmental features such as the Waikato river and lakes, and valuable dairy farmland, add further complexity.

Traditional preliminary mine planning techniques applied in these conditions are insufficient. Pit optimisation software, widely used in the metalliferous mining industry since the 1980s, offers the ability to analyse many more specific factors affecting pit limits, mine layouts, and economics, with more rigour and for a much larger number of scenarios, than traditional methods allow. SENZ therefore decided to adapt and use ‘Whittle’ pit optimisation software for Level 2 analysis of several Waikato opencast prospects. This paper describes this work.

INTRODUCTION

SENZ is New Zealand’s leading producer and distributor of high quality coals for export and domestic markets.

SENZ produces over four million tonnes of coal a year from its underground and opencast mines located near Huntly in the Waikato; Greymouth, Westport and Reefton on the West Coast; and Ohai in Southport, as depicted in Figure 1. More than half of the annual output is metallurgical coal, sold for export for use in steel production as well as the manufacture of carbon fibre, activated carbon and silicon metal.

SENZ’s two major domestic customers are the New Zealand Steel Ltd’s Glenbrook Steel Mill near Auckland, and the Genesis Energy’s Huntly Power Station. SENZ coal also supplies coal to the dairying, cement making, timber and meat processing industries throughout New Zealand.

Background

Demand for SENZ coal within New Zealand has increased in recent years, from 1.6 Mt in 2002, to a forecasted 3.0 Mt for 2005/2006. Over the next ten to 20 years coal-fired electricity plants could be required to provide between 500 and 1000 MW of new generation. This would create an anticipated additional demand for up to 3 Mtpa of coal.

SENZ uses a 20+ year period for integrated planning of all its business activities. It has an ongoing program of coal resource assessment to optimise future mines within this 20 year period, using a six step resource and mine planning process. Desktop review and conceptual study (Levels 1 and 2) are based on general coal resource information. Further geological and coal resource investigations are carried out for the more detailed evaluations in secondary assessment, feasibility study, feasibility study and detailed engineering (Levels 3 to 6).

As part of this program SENZ progressed planning for opencast options in the northern Waikato region to Level 2 (conceptual study). This required options to be analysed for a range of mine life – volume – cost scenarios. The complex geology, as well as the large number of potential pit targets and mining scenarios under consideration meant that an efficient method for both pit delineation and financial analysis was essential.

The metalliferous industry has for many years used pit optimisation software based on the Lerchs-Grossman algorithm (Lerchs and Grossman, 1965) to determine optimal pit limits and pit sequencing. Gemcom’s Whittle software was identified as the industry standard for pit optimisation, so after ensuring it met SENZ’s requirements, was utilised to carry out the task.

Although not commonplace, Whittle software has previously been used for coal applications. In the 1995 ‘Optimising with Whittle’ conference a paper was presented in which Whittle 3D software was used on a complex multi-seam coal deposit in Indonesia (Baafi, Milawarma and Cusak, 1995).
Current SENZ Waikato operations

Rotowaro opencasts

Rotowaro opencasts are located 10 km southwest of Huntly. Rotowaro coal was first mined in 1915 after a branch railway and a bridge over the Waikato River were completed. Opencast mining commenced in 1958. Currently there are two opencast mines in operation, Township (nearing completion) and Awaroa 4 (under development), producing approximately 1.6 Mt of coal annually. The majority of Rotowaro coal goes by rail to the New Zealand Steel Glenbrook steel mill and to the Genesis Energy Huntly Power Station. The remainder supplies North Island industrial and domestic markets.

At Rotowaro a coal washery and blending plant are used to process and blend coals to optimal product specifications. This enables SENZ to maximise the recovery and market value of its coal reserves.

Huntly East Underground Mine

Huntly East Underground Mine is located immediately north of Huntly township. The mine currently produces approximately 450 kt of coal per annum. Coal is loaded directly onto a branch railway at the mine, which connects to the nearby main trunk line. The majority of East Mine’s production is sold to New Zealand Steel’s Glenbrook steel mill.

Geology

The Waikato Coal Region consists of coalfields developed 30 - 35 million years ago (the Eocene-Oligocene period). The region extends from Drury (near Auckland) in the north to Mangapohue, south of Te Kuiti.

The Waikato Coal Measures were deposited on an eroded ‘basement’ of Mesozoic greywackes and argillites. The coal measures are overlain by a succession of marine sediments and volcanic deposits (three million years ago to recent time).

The general structural style is that of block faulting with steeply dipping normal faults.

For each of the Waikato coalfields under investigation, Vulcan grid models for both structure and coal quality existed. In order to utilise Whittle software these geology grid models required conversion to a three-dimensional block model. A common problem when converting grids to a block model occurs where coal and underburden are grouped together within the same block, potentially overstating mining costs for pit optimisation.

An in-house SENZ Vulcan macro was written to automate the model conversion, and to address the underburden problem.

Each block within the block model contains the following key information:

- recoverable tonnes and quality (ash, sulfur, specific energy) for each coal seam; and
- volumes and corresponding tonnages for each non-coal stratigraphic unit.

Note that spatial data for each block is inherent within the Vulcan block model framework.

Geotechnical parameters

In general, the stratigraphic horizons present can be consolidated into three geotechnical rocktype categories. Typical overall cut slope angles for each category are listed in Table 1.

Mining assumptions

As a first pass for the pit targeting exercise, it was assumed that targets would be large opencast operations, hence bulk earthmoving rates were used to determine the cost structure. Bucket wheel excavators and draglines were ruled out due to the high capital expenditure requirements, plus very weak waste material and unfavourable bedding dips. The mining method and base operating costs assumed were roughly based around the current Awaroa 4 operation, using large truck and shovel fleets (Komatsu PC4000 Excavator and 730E rear dump trucks) for waste.

Identification and evaluation of opencast targets

The process used by SENZ to identify and evaluate opencast potential in the Waikato Coalfields was as follows:

Stage 1 – Run Whittle Pit Shell Generator on coalfield-wide 3D block model

Generic geotechnical parameters were used in Whittle at this stage, differentiated by rock type category. Cut slope angles of 60° for basement, 30° for coal measures and 10° for ‘softs’ were used.

Bulk earthmoving truck and excavator fleet rates determined the unit cost structure. Coal revenue was defined by the specific energy content of the coal within the block model. A broad range of coal prices was used, from 50 per cent to 200 per cent of the base case revenue.

The potential underburden problem was addressed by converting all material below the target coal seam to air. This ensures that the costs and material quantities calculated during the pit optimisation process are accurate.

For each coal price analysed, Whittle Pit Shell Generator outputs the Lerchs-Grossman optimal pit outlines, and reports the corresponding quantities of coal and waste.

Any pit identified by Whittle at coal prices up to 200 per cent of the base case revenue was highlighted as a potential target area for follow up investigation. Conversely, areas where Whittle failed to identify a pit were eliminated from further consideration.

Stage 2 – Examine target areas

The resultant pit outlines were examined for several reasons:

1. The initial Whittle run was carried out on coalfield-wide models, which covers vast areas. Whittle performs more efficiently on smaller models, hence once target zones have been delineated, block model extents and block dimensions can be adjusted for further Whittle investigation.

2. To ensure that the pit dimensions are consistent with the cost inputs. If bulk earthmoving rates were applied, yet the pit outline infers that smaller mining fleets would be used, then the cost inputs need adjusting accordingly.

### Table 1

<table>
<thead>
<tr>
<th>Rock type category</th>
<th>Typical overall cut slope angle</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>Basement</td>
<td>60°+</td>
<td>Not generally mined, except where localised faulting disrupts stratigraphic sequence in adjacent blocks.</td>
</tr>
<tr>
<td>Coal measures</td>
<td>30 - 40°</td>
<td>Carbonaceous mudstones, shales and coal seams. Relatively competent compared to the overlying unit.</td>
</tr>
<tr>
<td>‘Softs’</td>
<td>8 - 15°</td>
<td>‘Softs’ is the colloquial term given to the mechanically incompetent weak rock and soils. In general this applies to all the material above the coal measures.</td>
</tr>
</tbody>
</table>

Not generally mined, except where localised faulting disrupts stratigraphic sequence in adjacent blocks.
3. To determine the location of the pit in relation to surface features. If the pit extends into areas beyond the control of SENZ (for legal, environmental, cultural, or infrastructure reasons), then either the block model requires modifying to identify exclusion zones, or additional costs need to be assigned in Whittle to fully cover the impact of mining (Wharton, 1996).

4. To determine whether the generic coalfield-wide geotechnical cut slope angles used are appropriate for the specific target area.

Stage 3 – Run Whittle Pit Shell Generator on each target area individually

Whittle Pit Shell Generator is run on the target area model, using adjusted geotechnical and cost inputs if required. The resultant pit outlines are re-examined. This is an iterative process, which continues until the assumptions used for the Whittle inputs, and pit outputs are in general agreement.

Stage 4 – Run Mining Scenario

Whittle Mining Scenario is run after defining mining rates, discount rates, and project capital expenditure requirements. A series of pushbacks can be manually or automatically defined within Whittle. Mining can be carried out either to balance quantity schedules or to defer waste, at the user’s discretion. For a given coal price, maximum project net present value (NPV) for each pit shell is computed. After validating the resulting Whittle production schedules to ensure mining practicality, coal tonnage and pit NPV data are exported into Microsoft Excel and graphed. The resulting price-tonnage-NPV graphs show the most profitable pit outline for any specific coal price, as well as the sensitivity of the particular target with respect to coal price and pit size.

This completes the Whittle characterisation, and provides extremely useful information for strategic decision-making.

Stage 5 – Detailed project evaluation

Selected Whittle pit shells as identified in Stage 4 are exported into Vulcan and used as the basis for detailed mine design, production scheduling, and financial analysis. By examining the price-tonnage-NPV curves, the mine designer has a good insight into the sensitivity of the pit economics and pit geometry to changes in coal price. Armed with this information, pit designs can to some extent be ‘future proofed’ to allow for possible expansion or contraction, depending on prevailing economic conditions.

CASE STUDY – KIMIHIя OPENCAST EXPANSION

As part of the Huntly Coalfield regional assessment, the Whittle Pit Shell Generator identified a potential pit target immediately adjacent to the old Kimihia opencast mine.

Background

The existing Kimihia opencast pit was mined from 1944 to 1977. The portals of Huntly East underground mine are located in the western wall of the old pit. Settling ponds and other infrastructure servicing the Huntly East Mine are located within, and adjacent to the former pit. Just prior to the commencement of the Waikato coal targeting exercise, the Kimihia pit area had been the subject of a traditional mine design conceptual study. This was seen as an excellent opportunity to validate and compare the Whittle and traditional process and outputs.

Geology

In the prospect area, the Kupakupa Seam, ranging from 3 to 8 m of coal (excluding partings), is overlain by the Renown seam, ranging from 2 to 4 m of coal (excluding partings), with up to 30 m of interburden between the seams.

The Kimihia target area is typical of the Waikato coalfields, with coal measures overlain by younger sedimentary formations classified as ‘softs’. Softs account for approximately half the thickness of non-coal strata within the target area.

The coal in this area consists of multiple split Renown and Kupakupa seams, typically medium ash (~10 per cent) coal with specific energy of 22 MJ/kg (as received basis), and total sulfur of 0.3 per cent.

Geotechnical

The pit design parameters used for the Whittle Pit Shell Generator were modified from the generic Waikato-wide cut slope angles to account for local conditions. Overall slope angles of 60° for basement, 34° for coal measures, and 11° for ‘softs’ were chosen.

Mining

The coal quality within the target area meant that Kimihia coal would require blending with other coal sources. For the purpose of this study, it was assumed that the coal production from the target area was limited to 500 kt per annum.

The characteristics and size of the target meant that waste removal by truck/shovel operation using Komatsu PC4000 excavators with Komatsu 730E trucks, or equivalent was considered appropriate.

Two methods of coal winning were assessed:

1. bulk winning of coal and included partings, followed by processing in a dense medium washery; or
2. selective coal winning using 45 tonne excavators on dayshift only, in order to minimise contamination during selective mining of thin coal seams.

The selective option was favoured after economic analysis showed the costs of a dense medium plant to be prohibitive, hence coal mining costs for Whittle optimisation were based on selective mining using 45 t excavators with 40 t trucks.

Surface features

Within the Kimihia target area there exist a number of infrastructure and other surface features, including public roads, Huntly East Mine site infrastructure and underground access roadways. In addition, a proposed diversion of State Highway One in the northeast of the target area is scheduled for construction by 2020.

Due to the large number of surface features, two separate scenarios were considered. Firstly the model was constrained to avoid all surface infrastructure, however constraining the Whittle Pit Shell Generator in this manner meant that no economic pit could be found.

The second scenario did not constrain the Whittle Pit Shell Generator in any way, however the additional costs associated with relocating or replacing each surface feature were built in to the model to reflect the associated financial implications.

Whittle outputs

The Whittle Pit Shell Generator was re-run using the revised Kimihia-specific parameters. Next, the Whittle Mining Scenario program was run to produce mine schedules that maximise project NPV based on 500 kt per annum coal production, while balancing annual waste production, for a specified range of coal prices.
Whittle Mining Scenario results were exported into Microsoft Excel, and subsequently graphed. Figure 2 shows an example of the graph obtained from this process. Note that the results shown within this paper have been modified to protect commercially sensitive information.

The base case revenue curve (rev1.0), represented in Figure 2 shows a positive project NPV for pit sizes up to and including approximately 2.6 Mt, with a maximum NPV achieved at approximately 2.4 Mt. Note that for the range of pit sizes from approximately 1.9 - 2.6 Mt there is very little change in project NPV. This infers that when the detailed pit design is undertaken, minor deviations from the optimal Whittle pit outline will not cause a significant change in project economics.

At coal prices ten per cent greater than the base case (represented by the rev1.1 curve), Figure 2 indicates that the project economics could support a larger pit, up to 6.5 Mt and achieve a positive NPV. Note that the dashed lines between the 2.6 Mt and 6.4 Mt pits represents a significant jump in pit size, and is due to a down thrown faulted block of coal.

In this specific example, if maximising project NPV is the company objective, there is no point in chasing the larger pit options, as under all scenarios the highest project NPV is achieved at pit sizes of 2.6 Mt or less.

If the 2.4 Mt pit option is chosen, representing the maximum project NPV for the base case coal price, it can also be seen that in the event of coal prices dropping 20 per cent (represented by the rev0.8 curve), the project still achieves a positive NPV. If the coal price drops 40 per cent below the base case (represented by the rev0.6 curve), then the project never achieves a positive NPV under any pit size.

In order to ‘future-proof’ the pit project, the mine designer would be wise to avoid sterilising the coal between the 2.4 Mt and 2.6 Mt pit options, as there is some potential upside in the event that the coal price increases above the base case during the project life. There is no benefit in avoiding sterilisation of the coal beyond the 2.6 Mt pit limits, unless practicable to do so without incurring additional mining costs, as under no scenario does the larger pit option appear financially attractive compared to the smaller pit options.

All this is extremely valuable information to have at the time of mine design and initial project assessment.

**Whittle versus traditional method**

Prior to using Whittle, the preferred mining scenario for Kimihia, as determined by traditional methods, was a pit containing 2.4 Mt of mineable coal. A number of pit options, using vertical strip ratio and surface constraints as a guide, were designed in Vulcan and analysed individually before determining the preferred scenario.
Using Whittle, numerous mining scenarios were examined, in which costs, cut slope angles, production rates, coal prices, and exclusion zones were varied. The entire process only took two days. The resulting preferred Whittle pit option also contained 2.4 Mt of mineable coal, however it did so with significantly less overburden material than the traditional pit, and hence achieved a more favourable financial outcome.

The results achieved using Whittle on the Kimihia target area validated the methodology, as well as giving credibility to its use within SENZ. The high level of structural complexity at Kimihia resulted in the Whittle method outperforming the traditional mine design method, and doing so in a fraction of the time.

**Future work**

Results from the Whittle targeting exercise have shown a potential economic target, albeit a fairly small tonnage project, in the Kimihia region of Huntly coalfield.

Limitations in the geological model for the Kimihia target area have been identified, and a resource definition drilling program has commenced. A more extensive Whittle investigation will take place once the updated geological model is available, before progressing the project to a full pre-feasibility study.

**CONCLUDING REMARKS**

Use of Whittle pit optimisation software for evaluation of opencast options in the Waikato coalfields has set a new benchmark for SENZ project assessment methodology for geologically complex resources at Conceptual Study stage. SENZ opencast projects are now being routinely analysed using Whittle as part of the strategic mine planning process.

**REFERENCES**


Coming of Age for Low-Density Explosives

J Rock¹, A Maurer² and N Pereira³

ABSTRACT
Low-density explosives have been developed with trials being conducted for over 20 years and yet have still only gained limited market acceptance despite producing some very promising results. The biggest concern with industry acceptance has been disbelief that a product with lower density than the ANFO benchmark could fragment anything but the weakest of strata. Trials have been carried out with products such as diluted ANFO, low-density ammonium nitrate (AN) and various other mixes, although it has been only recently that low-density explosives have been accepted as a serious alternative to traditional products such as ANFO and heavy ANFO.

The recent uptake of low-density products has been the result of several key factors:
- the current resources boom forcing AN supply issues,
- the development of new handling techniques, and
- a far better understanding of the utilisation of such products.

This paper highlights the benefits of low-density products available in the market and focuses on the situations where low-density explosives can provide the end user with benefits that would otherwise be achieved through more time consuming methods.

Through correct implementation, low-density explosives can provide the blast designer with another option when looking at the optimum method of breaking the rock. This while still controlling the other limiting factors such as cost, environmental impacts (noise, vibration, dust and fumes), coal damage and safety (stable walls).

INTRODUCTION
Low-density explosives (LDE) have had a long time in development, with early studies going back to the 1970s. These early investigations focused on reducing the density of the commonly used ANFO product. The widely accepted line of thinking was that ANFO was the lowest density product available at the time and as such, this would be the ideal starting point from which to construct a lower density product. This has led to many different researchers investigating the characteristics of various formulations to try to arrive at a product that would not only be reproducible in the field, but also be commercially competitive with industry accepted products.

While several LDE products have been commercially available for many years, it has only been in recent years that they have been looked at as a viable alternative to ANFO for all but specialised applications. The main hurdle for LDE has been overcoming the perception that it is not possible to break rock with considerably lower powder factors than used with ANFO.

BACKGROUND
Although studies have continually stated the positive benefits of using LDE, the uptake from industry has been slow – partly because of fear of an unknown product, and a perception that LDE were only suitable for ‘weak strata.’ The main use for LDE has been for niche applications. One of the other main sticking points has been the large-scale handling of the low-density bulking agent. This has effectively led to an impression that although the results are promising, the ongoing use of the product has been put in the ‘too hard’ basket by mine operators.

If the use of AN as the primary raw material in an explosives is examined, it too took a while to gain acceptance as a standard product. Ammonium nitrate was first discovered as an explosive ingredient when mixed in with nitroglycerine by Swedish chemists Ohlssen and Norrbin in 1867. Literature also makes mention of its ability to form an explosive when mixed with a hydrocarbon. McAdam and Westwater (1958) document how Alfred Nobel realised the potential value of this type of explosive and acquired the patent rights from his fellow countrymen. These early explosives used AN as an ingredient to be combined with nitroglycerine. However, it wasn’t until the Texas City explosion in April 1947, along with other developments in the use of AN, that AN as the primary ingredient started being considered a viable explosive in its own right. Hopler (1993) makes mention of the low cost of AN following the end of WWII during which ten ammonium plants were built for the munitions industry to support the war. This, combined with drilling technology that allowed large diameter holes to be drilled rapidly and cheaply, called for an effective explosive product that could be loaded quickly and easily. By mid-1956, ammonium nitrate was being mixed with fuel oil (diesel), and poured from the bag into the drill hole.

Early references to LDE date back to the late 1960s with the Blasters’ Handbook from Du Pont (1969) referring to a Du Pont product named ‘Nilite ND’ with a density range from 0.45 g/cc to 0.55 g/cc as poured (the ND meaning ‘no-deck’). This product:

… has proven successful in vertical holes when it has been used as a top load. It has successfully replaced decking in quarry shots and is used where the total charge per borehole must be kept below a maximum weight.

IRECO had its version of a low-density product available named Isanol, which was essentially an ANFO/polystyrene mixture.

During the 1970s, several investigations into the use of LDE were conducted, in particular the use of Isanole. These culminated in a paper by Heltzen and Kure (1980), which showed that a low-density product could be mixed with minimal segregation that was very effective for contour blasting. The main drawback was the additional handling costs associated with the product if there was only going to be minimal application. However, this study did highlight that an effective low-density product could be delivered that sustained minimal segregation along with no static effects and similar CO and NO outputs.

Wilson and Moxon (1989) conducted extensive trials diluting ANFO with various low-density bulking agents including polystyrene, bagasse (sugar cane waste) and sawdust. The main aim of these trials was to ‘… develop a low-shock energy ammonium nitrate based explosive which could be used to fragment weak overburden materials’. They found that ANFO diluted with different products could be easily mixed, could be made homogeneous and had detonation pressures that could be controlled. The final point from this study however, was that ‘… low-density explosives can lead to significant cost savings without compromising fragmentation or production’.

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In the early 1990s LDE received more attention with Hunter, Fedak and Todeseschuck (1993) investigating the use of an ANFO based LDE in wall control applications. This study looked at a range of densities from 0.36 g/cc through to 0.45 g/cc with a view to reducing ore dilution and minimising damage to the final wall. Other techniques considered at the time were pre-splitting, line drilling and decoupling of charges but were ruled out due to the irregular nature of the geology and the size of the blastholes. The other requirement of this product was its ability to be pneumatically loaded. The result of these trials was a low-density product that could be loaded consistently, that performed reliably and resulted in lower levels of blast induced damage and vibration.

At a similar time Jackson (1993) was undertaking field trials of an emulsion based LDE that was a combination of chemical gassing agents, glass micro balloons and polystyrene beads. Various mixtures were tested to determine the most easily field reproducible as well as the most economically viable. The results from these trials found that powder factors could be reduced by as much as 30 per cent, while at the same time producing similar results in terms of fragmentation, breakeage, better wall stability and reduced fines. Again, this provided evidence of very good results from the utilisation of lower density products in areas that would traditionally be blasted with ANFO.

This led to a study by Grouhel and Hunsaker (1995) undertaken in the Hunter Valley region of NSW to determine the viability of LDE products similar to that investigated by Jackson. This study found that similar results to that achieved by ANFO could be produced with a low-density product at a density of 0.6 g/cc. The LDE product trialled was deemed a suitable alternative for the majority of medium to large diameter blastholes (>150 mm) where ANFO was the explosive commonly used.

Several trials were conducted in the Bowen Basin with limited acceptance of an ANFO/sawdust mix as documented by Johnson (1996). This saw reasonable quantities of product utilised to replace ANFO in softer geologies. It described significant cost savings over ANFO while providing comparable results. While this process found some acceptance, its use was limited.

Brent and Armstrong (1998) conducted trials primarily looking at the application of LDE for pre-split applications. Using a very low-density product (0.2 g/cc) in large diameter blastholes (311 mm) at depths of 45 m, the half barrel factor was increased from 32 per cent to 62 per cent purely by having a better distribution of the product in the hole (half barrel factor is the percentage of the blasthole visible following excavation of the shot material). Again this showed the benefits of using LDE products in an ongoing application, but without a significant driver behind its use (a clear benefit to the site being economic, safety or environmental), it was relegated to the trial status.

Rowe et al (2001) conducted a study with a variable density product to determine its suitability in soft to medium strength rock types. The primary focus was on the ability to load a lower density product into holes regardless of moisture that didn’t require blasting with higher density products. They found a range of products that could be tailored to ground conditions to provide the blast designer with a customised system of explosive delivery without compromising results. This has subsequently been finding gradual market acceptance as industry has gained a better understanding of the utilisation of lower density products. This product has achieved greater success due to its ability to utilise current on-bench equipment (Mobile Manufacturing Unit – MMU) without having additional or purpose built delivery trucks.

Further work on an ANFO based LDE was conducted by Beach et al (2004) using wheat husks as the bulking agent with an ANFO base. While this paper reported good results in terms of the LDE employed, it required specialised handling equipment and in the words of the authors ‘...it is suitable for blasting weak ground with dry holes’ and ‘...is suited to weak strata’. Again, this shows that although LDE can be utilised successfully in a large-scale blasting scenario, without significant investment in dedicated equipment, the ongoing use is limited. At the same time Rock (2004) prepared a paper on the merits of LDE based on a bulked out emulsion-based product. This paper highlighted the strengths of lower density products and the techniques to use when designing blasts for such products. It also put forward some of the theory behind the success of LDE and why it works when conventional thinking says it should not.

As highlighted in several of the papers above for a low-density explosive to be viable both operationally and economically, it needs to demonstrate several main characteristics:

- low-density bulking agent – to reduce the density of the product being diluted the bulking agent is ideally lower than 0.15 g/cc;
- ease of handling – product needs to be as easy to load into the blasthole as the higher density product it is replacing;
- non-segregating – product must be homogeneous when loaded into the blasthole (and not segregate while loading);
- equivalent load rate – equipment must be able to reload and load the same number of blastholes as the product it is replacing; and
- lower cost – the use of a LDE must provide an economic or tangible advantage (such as lower vibration or less caprock) to encourage the mine operator to use the product.

**PARTITION OF ENERGY**

There have been several research papers on the breakage process due to explosives. When an explosive detonates in a blasthole, the sudden and rapid release of energy produces very high pressures which initiate a fracture network around the blasthole. As this network expands, the pressure in the blasthole subsequently reduces according to the P-V relationship applicable for that explosive in that rock type. Singh (1999) proposed that although much of the energy in AN based explosives is interpreted as heave energy, the utilisation effectiveness is dependent upon the pre-conditioning of the rock and the extent of the fracture network created by the early stages of energy release and pressure application. A simple, idealistic, static energy release model has been proposed by Lownds (1986) in which the zones are partitioned into the commonly known components – shock, heave and losses. As can be seen in Figure 1, the pressure following detonation rapidly drops off as the explosive expands.

The areas and points in Figure 1 are represented thus:

- potential shock energy – area 1,
- strain energy around the borehole – area 2,
- fragmentation and heave – area 3a,
- strain energy in burden at escape – area 3b,
- lost energy – area 4,
- initial detonation pressure – point P1,
- pressure at end of shock phase – point P2, and
- pressure after which no further work is done on the rock – point P3 (cut-off pressure, usually 100 MPa).

As rock is a brittle material, it will break far more effectively in tension than in compression. In the early stages of energy release, some energy is expended crushing and fracturing the area immediately surrounding the blasthole. Energy is also utilised initiating and extending the predominantly radial fracture network away from the blasthole. Energy is then expended opening up both the natural joints and cracks in the rock mass as well as the fractures developed by the earlier high pressures prior to the bulk motion or heave which is manifest as kinetic energy.
So whereas conventional thinking has stipulated that higher VOD (and subsequently higher pressure) products produce better results in all but weak strata, this only really holds true for truly massive rock formations with minimal and irregular joint and micro-cracking (such as massive granites). In rock types that display jointing and inherent cracking (such as that found in the majority of coal mining overburden), the requirement for high initial pressures is minimal. As such a more optimal blast in terms of the correct energy for the rock can be provided by products that display significant partitioning towards heave (gas or bubble energy). Figure 2 displays the tapering off of the stress in rock versus volume due to the compression and crushing around the blasthole. This tapering off of the stress in rock is caused by the initial compression and crushing around the blasthole followed by growth of the fracture network and then the movement of the rock mass. The actual interaction point is further along the expansion curve than if it were a purely elastic reaction.

Once this interaction point is reached, the heave phase of the process takes over and further fragmentation and breakage is caused by this movement of the rock. As low-density products have a lower VOD, the explosion expansion curve has a lower starting pressure. This lower VOD and initial pressure translates into an increased percentage of the available energy applied during the heave process. A low-density product will still utilise some of its available energy during the initial expansion process, however this is a smaller percentage when compared to ANFO or higher density products.

It should be noted that models such as that proposed by Lownds do not account for the dynamics of blasting. More important than volume expansion, is the rate at which explosive energy is delivered to the rock as this not only controls the stress or strain in the rock, but also the strain rate which can profoundly affect the crack initiation and ultimate fracturing particularly in the near field. Higher strain rates generally lead to more fracturing and smaller fragmentation. Despite having lower detonation pressures, the energy release rate of low-density explosives is very similar to that of ANFO.

**CASE STUDIES**

**Bengalla Mine**

Bengalla Mine is managed by Coal and Allied on behalf of Rio Tinto Coal and its partners. It is a low-cost operation in the Hunter Valley producing six million tonnes of coal in 2003 and blasting 19 million bank cubic metres of material. Bengalla is 1.5 km from the township of Muswellbrook and is surrounded by a number of residences as shown in Figure 3. The management of environmental effects including blast induced fumes on its surroundings is of paramount importance to the mine.

Historically the mine has used ANFO explosives. However, due to the inherent moisture within its clay based overburden material, suboptimal detonation of ANFO has resulted. If slept for more than 24 hours the deterioration would produce post blast fumes. As a result of this, hole liners were employed to reduce the effect of ANFO deterioration. Although the combination of ANFO and hole liners were cost-effective compared to other conventional bulk emulsion based explosives, issues of reduced on-bench loading productivity and twisted liners within the blasthole resulting in bridged loading were still a concern. There was also a relationship with ANFO and hole liners that had been slept greater than seven days still producing fumes.

The low-density product Flexigel™ CLEAR was highlighted as a suitable replacement product for ANFO and hole liners, to reduce both blasting costs and eliminate fume generation. A series of blasts were conducted between March and September 2004 to explore and demonstrate the suitability of this product at Bengalla Coal Mine. During this period a total of seven shots, 1600 blastholes and 600 tonnes of low-density product were fired successfully. The low-density explosive was used in both partly loaded shots to compare directly against ANFO and hole liners.
as well as fully loaded shots once confidence in the product was established. Results clearly indicated that Flexigel™ CLEAR did not fume in situations where conventional bulk explosives normally do. In addition to fume reduction Bengalla experienced reduced blast induced vibration and dust generation attributed to the low-density.

With a change in the mining sequence the mine wanted to take advantage of reducing their overburden inventory through the removal of buffer material and to fire their shots with a free face. The buffer material is shot material from the previous blast that slows and dampens the movement of the material thus reducing airblast. A trial blast was fired without the usual safety net of buffer material in front of the shot and the airblast results came very close to exceeding when using ANFO in the front row of holes. Through an investigation involving mine personnel and Orica Technical Services personnel, a design procedure was implemented to reduce the risk of an exceedance.

This investigation highlighted that where conventional explosive products were used, the front row burdens were not sufficient to control the face. This caused a high acceleration rate of the face material, which resulted in an airblast recording close to the site limit.

A design procedure was put in place to reduce the risk of drilling front row burdens that could not be controlled. This involved laser profiling and creating cross sections through every front row hole to ensure that every hole had sufficient burden. However, it was found that with all this in place some holes were still drilled too close to the free face with some burdens being only 2 m. This meant an alternative course of action was required to be incorporated into the procedure to account for small burdens that had already been drilled and were less than design.

When a cross section pinpointed a blasthole or blastholes that had less than the design burden, 0.5 g/cc density Flexigel™ replaced the standard product that was used. Due to the lower Velocity of Detonation (VOD) of Flexigel™ compared with ANFO/other bulk explosives, the face moves at a lower velocity and therefore reduces the chance of exceeding airblast limits.

**Hunter Valley Operations**

Hunter Valley Operations is an amalgamation of the Howick, Hunter Valley and Lemington mines and is managed by Rio Tinto Coal Australia. The operations produce a total of 12 Mt of domestic and export steaming coal as well as semi-soft coking coal. While the operations are approximately halfway between the townships of Singleton and Muswellbrook, there are still requirements for the blasting program to stay within environmental limits.

As the overburden removal process progresses, blasting adjacent to high voltage overhead powerlines presented dragline scheduling issues due to the size of the shots required and vibration issues. The mining sequence was bringing the shots closer to the powerlines every strip and due to the dip of the seam the depth of material to be blasted was increasing. This resulted in deeper blastholes requiring increased charge weights that limited blast size in order to control ground vibration.

Through the application of 0.5 g/cc density Flexigel™, the blast lengths were increased from 100 m to 200 m in length while still maintaining the charge weight per delay requirement stipulated by the site. Ground vibration was required to stay below 50 mm/s at he power line towers (150 m at the closest point) to comply with the mining lease conditions. The results from the blast halved the vibration levels compared to previous shots recording 25 mm/s which were the lowest levels recorded in that area.

**Newlands Coal Mine**

Newlands Coal Mine, owned and managed by Xstrata Coal, is in the northern part of Queensland’s Bowen Basin, 130 km west of Mackay, and 32 km north-west of Glenden. The coal is a high quality, medium volatile, low sulfur steaming coal that varies between 5.5 and 6.5 m thickness. It is mined from underground and open-cut operations that together produce seven million tonnes of washed coal per year.

In 2004, Newlands opened its Suttor Creek deposit, mining 16 million bank cubic metres to uncover a total 2.7 million tonnes of coal annually. The overburden material consists of a combination of claystone and siltstone and is mined using a BE1370 dragline fitted with a 37 cubic metre scoop bucket. Blasted material must be both well fragmented and loose to provide optimum digging performance. Historically the mine has blasted this material using ANFO, with Energan 13 (density 1.3 g/cc) toe charges. Blasting in these conditions produced significant post blast fumes. The Suttor Creek deposit was highlighted as a suitable pit for using low-density explosives and in particular Flexigel™ CLEAR to replace ANFO and Energan explosives to reduce both blasting costs and fume generation.

Blasting with Flexigel™ low-density explosives began in Suttor Creek in September 2004. The mine has reported that the blasted material presented to the dragline has been excellent, dragline productivity was very high for this machine achieving between 50 - 60 thousand bcm per day showing low wear rates on the bucket.

**BENEFITS OF LOW-DENSITY PRODUCTS**

Based on previous work published by various authors, low-density explosives have been utilised in the following situations:

- **Reduction of toe in coal mining** – being able to drill every blasthole to coal or reduce coal stand-off to ensure all toe material is blasted.
- **Protection of coal** – minimal shock energy prevents the product from damaging the coal roof.
- **Reduction of caprock** – lower VOD allows stemming to be reduced, thus placing product in the zone where cap material is produced.
- **Protection of walls** – using lower VOD products in pre-split can improve wall conditions considerably. Reduced back break allows some pre-splitting to be eliminated altogether.
- **Management of environmental results** – in side by side comparisons with ANFO, low-density products have reduced both vibration and overpressure by around 30 per cent.
- **Cost benefits** – by utilising a lower density product, fewer kilograms of explosives are loaded into the blasthole.
- **Reduction of fines** – because low-density product is heavily partitioned away from shock energy, the area around the blasthole that is normally crushed is reduced.
- **Absence of fumes** – formulation and lower VOD ensures complete consumption of all components within the blasthole, fumes have been all but eliminated with Flexigel™ CLEAR.
- **Density on demand** – a continuous mix product such as the Flexigel™ range, has the ability to vary the density being loaded into the hole to allow for changing rock types.
- **Moisture resistance** – A high emulsion content product such as Flexigel™, has the ability to withstand moisture where ANFO based low-density products cannot.
DISCUSSION

If the uptake of low-density products is looked at closely, there are several factors that have contributed to its recent increased use:

- Increased understanding of application – with knowledge of the product and how to apply its unique characteristics, the product can be utilised to great advantage.
- Increased investment in capital – with both mining companies and explosives supply companies showing keen interest in the development of this product, capital has been invested to provide the opportunity to deliver at sustainable production rates.
- Current market trends – with current resources upswing, the supply of raw materials have become increasingly stretched. This has led the industry to look for other options to maintain and in many cases increase production. The use of low-density products has largely focused on ground that has typically been blasted with ANFO.
- The time between the initial development and large-scale application of LDE has taken some 40 years. If this is compared with the initial recognition of ammonium nitrate as a blasting agent and its application, which took nearly 80 years, the use of LDE has rapidly gained acceptance.

CONCLUSIONS

Low-density explosives such as Flexigel™, offer explosive users an opportunity to enhance their current selection of blasting products. The advantages of reduced product consumption and increased control are significant in themselves. Combine this with moisture resistance and density on demand LDE such as Flexigel™, provide the blast designer with a product that can be used to complement more established products.

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R70 Relationships and Their Interpretation at a Mine Site
B B Beamish¹, D G Blazak², L C S Hogarth³ and I Jabouri⁴

ABSTRACT
The R70 test is a good indicator of coal reactivity to oxygen and is commonly used to provide a rating of propensity for self-heating. Tests performed by The University of Queensland Spontaneous Combustion Testing Laboratory have resulted in the development of a large database that shows the impact of coal quality on this parameter. From these results it is possible to infer an R70 value, which can be used to gain an appreciation of the intrinsic coal reactivity before it is even tested. However, any interpretation of R70 values must consider the influences of coal rank, mineral matter and moisture content to provide a better understanding of the spontaneous combustion risk of a particular coal.

INTRODUCTION
The self-heating of coal is due to a number of complex exothermic reactions. Coal will continue to self-heat provided that there is a continuous air supply and the heat produced is not dissipated. The parameters that control a coal’s propensity for self-heating have been the subject of many investigations. Relationships between coal properties and self-heating indices have been published in a number of studies (Humphreys, Rowlands and Cudmore, 1981; Beamish, Barakat and St George, 2000, 2001; Moxon and Richardson, 1985; Singh and Demirbilek, 1987; Barve and Mahadevan, 1994).

Humphreys, Rowlands and Cudmore (1981) found a simple relationship between the coal self-heating index parameter, R70, and coal rank. However, work by Beamish, Barakat and St. George (2001) on New Zealand coals covering a wider range of coal ranks showed that the rank relationship with R70 coal self-heating rate is non-linear. Beamish and Blazak (in press) show that R70 values decrease significantly with increasing mineral matter content, as defined by the ash content of the coal. This paper presents results from the large R70 database that has been developed at The University of Queensland for coals from both Australia and New Zealand. These results show coal quality trends that can be used to infer R70 values for coals with no previous testing history. Furthermore, a discussion is presented on the possible interpretations that need to be considered when using this parameter to evaluate the propensity of a coal to self-heat.

COAL SAMPLES AND R70 TESTING PROCEDURE
The coal samples referred to in this paper are test results from published (Beamish, Barakat and St George, 2001; Beamish, in press; Beamish and Blazak, in press) and unpublished (Hogarth, 2003; Jabouri, 2004) studies on coal self-heating. The samples range in rank from subbituminous to medium volatile bituminous and cover a wide ash content range from 0.7 to 63.9 per cent, dry basis. Several New Zealand coal regions are included in the database. These are Waikato, Reefton and Greymouth. Coals from both the Bowen and Sydney Basins have also been included.

The testing procedure essentially involves drying a 150 g sample of <212 mm crushed coal at 110°C under nitrogen for approximately 16 hours. Whilst still under nitrogen, the coal is cooled to 40°C before being transferred to an adiabatic oven. Once the coal temperature has equilibrated at 40°C under a nitrogen flow in the adiabatic oven, oxygen is passed through the sample at 50 ml/min. A data logger records the temperature rise due to the self-heating of the coal. The average rate that the coal temperature rises between 40°C and 70°C is the self-heating rate index (R70), which is in units of °C/h and is a good indicator of the intrinsic coal reactivity towards oxygen.

RELATIONSHIP BETWEEN COAL RANK, ASH CONTENT AND R70 VALUES
As the R70 value is obtained on a dry basis, the best way to graphically represent the data is to plot it against the ash content (on a dry basis, Figure 1), which is a standard analytical determination for coal. The ash content is closely related to the mineral matter in the coal, which is the inorganic constituents of the coal that modify the coal behaviour in many combustion processes. In the case of the coal self-heating, the mineral matter acts as a diluent.

Smith, Miron and Lazzara (1988) discussed the thermal effects of additional mineral matter in coal and pointed out that, assuming the additive is inert to the oxidiser, the additive acts as a heat sink. Consequently, the reaction rate is lowered, reducing the self-heating propensity of the coal. This is clearly seen in Figure 1 for the Trap Gully, Dunn Creek, Newcastle and Bowen Basin coals. Humphreys, Rowlands and Cudmore (1981) also proved this effect by adding ash to coals to test an equation for mineral matter-free correction of the R70 value. In the present study, a mineral matter-free correction is not necessary, as the pure-coal R70 value can be obtained from the y-intercept of the trendline equations for isorank coal from the one seam.

Figure 1 shows that the subbituminous coals have the highest R70 values for any given ash content. There also appears to be no major difference between the Waikato subbituminous coals and the Trap Gully subbituminous coal, this is despite there being a substantial difference in the maceral composition between the two. The Waikato coals contain very little inertinite, whereas the Trap Gully coal contains a significant amount of inertinite. This is somewhat surprising and may be an artefact of the type of sample tested. The Trap Gully samples were all fresh cores that were firstly wrapped in plastic cling wrap then aluminium foil and an outer layer of masking tape before being frozen on-site. The cores were then transported to The University of Queensland in an insulated container full of ice. On arrival, the cores were transferred to a freezer for storage until adiabatic testing took place. These precautions were taken to preserve the intact core and minimise pre-oxidation effects before testing. However, the Waikato samples were supplied from the Coal Research Ltd sample bank and may have undergone some oxidation prior to testing.

The Reefton subbituminous coal has a much lower R70 value compared with the Waikato coals. This may be due to the coal having a lower rank. The Reefton high volatile bituminous coal is at the lower end of the high volatile bituminous rank range and appears to have a lower R70 value than expected. One reason for this may be the presence of a significant amount of organically-bound sulfur in the coal, which presumably blocks access of oxygen to reaction sites.
The rank and ash relationship shown in Figure 1 makes it possible to infer a reasonable value for R70 based on coal quality in areas where no test information is available. However, it should be noted that subtle differences in coal reactivity can occur due to the different types of mineral matter that are present. Consequently, in any final mine assessment there is no substitute for hard data.

**INTERPRETING R70 VALUES IN TERMS OF PROPENSITY FOR SPONTANEOUS COMBUSTION**

Interpreting the significance of the R70 value has often been problematical for mining operations, particularly as there is a wider range of coals being mined than when the test was first developed. This is most likely a function of the test procedure, which has a common start temperature (40°C), uses dry pulverised coal (<212 mm) and uses oxygen as the reactant at a high flowrate/mass ratio. Nevertheless, knowing the intrinsic reactivity of the coal is a good starting point to assess the propensity of a coal for spontaneous combustion.

It can be seen from Figure 1, that a low rank high ash content coal can have the same R70 value as a high rank lower ash content coal. The R70 curves for two coals that fit this criterion are presented in Figure 2. The graph clearly shows that even though both coals have the same R70, the coal with the lower ash content reaches thermal runaway some six hours earlier. This is a function of the difference in heat capacities of the two coals and the heat of oxidation from the reaction. This is an important difference that must be considered when comparing coals from different mines.

The modifying influences of other factors also need to be considered when assessing the meaning of an R70 value. For example, Beamish and Hamilton (in press) have shown that the accelerated effect of coal reactivity does not take place until the moisture content of the coal drops to approximately 50 per cent of the moisture holding capacity. This is due to the competing influences of heat loss through evaporation and blocking of access to oxidation sites by the moisture. Figure 3 shows this effect for a subbituminous coal tested using the R70 adiabatic oven. The sample at zero per cent moisture is the normal R70 curve, but when the coal was tested at a moisture content of 9.9 per cent, the self-heating period was extended considerably, before accelerated oxidation took place. What influence does this effect have on spontaneous combustion propensity interpretation?

The Trap Gully sample with an R70 of 16.22°C/h has an ash content of 9.8 per cent db and a moisture holding capacity of approximately 14 per cent. The Trap Gully sample with an R70 of 1.69°C/h has an ash content of 62.2 per cent db and a moisture holding capacity of nine per cent. Consequently, the higher R70 sample has to lose seven per cent moisture before the coal reactivity can lead to intense oxidation, whereas the lower R70 sample only has to lose 4.5 per cent moisture to reach the same point. This 2.5 per cent moisture differential has a significant impact.

The issues of moisture and moisture movement are complex in that removal of moisture can lead to changes in coal structure that increase the available surface area for oxidation. The adsorption of moisture itself leads to heat generation, which will increase the rate of heating. These issues are best examined in large test environments.

It is apparent from comparison of the Trap Gully samples that even high ash rejects have the potential to self-heat in a relatively short period of time. Similarly, poor quality coal, on the basis of high ash content, that is left behind in a goaf area still has the potential to pose a spontaneous combustion risk.

**CONCLUSIONS**

The intrinsic reactivity of coal towards oxygen can be accurately measured by the R70 test procedure. The value of this parameter is strongly affected by rank and mineral matter. In addition, the moderating effects of moisture in the as-mined coal are not taken into consideration by this parameter, and hence there is a need to review the way in which this and other small-scale index parameters such as crossing point temperature (CPT) and minimum self-heating temperature (SHTmin) are used in spontaneous combustion risk assessments.

One possible solution is to use numerical modelling to scale up the results and incorporate the missing influencing factors. However, there are dangers of oversimplification of the coal self-heating processes when incorporating only small-scale data into the models. A more practical approach is to obtain results from bulk testing of coal using conditions closer to the as-mined situation and history match this data with a more robust numerical model that can incorporate the effects of moisture in particular.

Preliminary work is being performed as part of ACARP project C12018 to establish if this is possible, and comparisons with the R70 data are being made to provide guidelines for optimal strategies of spontaneous combustion risk assessment.
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Factors Affecting Hot Spot Development in Bulk Coal and Associated Gas Evolution

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ABSTRACT
There are many physical factors that can affect the self-heating rate of coal. The presence of seam gas has often been referred to as inhibiting coal self-heating due to the limited access of oxidation sites created by the presence of the gas adsorbed on the coal pores. Similarly, the presence of bed moisture in the coal also acts as an inhibitor of oxidation by blocking access of air into the pores. Gas drainage of a coal seam prior to mining removes both gas and moisture from the seam. Bulk coal self-heating tests in a two-metre column on both gassy, as-mined and gas-drained, dried high volatile bituminous coal show that removal of gas and moisture from the coal accelerates the rate of self-heating to thermal runaway from 8.5 days to 4.25 days, from a start temperature of 30°C, with an airflow of 0.25 L/min. The corresponding gas evolution pattern for each of these situations is different. Therefore, it is necessary to take this change in coal condition into consideration when developing a spontaneous combustion management plan.

INTRODUCTION
Self-heating leading to spontaneous combustion continues to pose a significant hazard during the mining of coal. A recent example of this is Southland Colliery in December 2003, where a heating progressed to ignition forcing the mine to be closed. Unfortunately, the heterogeneous nature of coal and the contributing factors that control whether heat is gained or lost from the coal/oxygen system make it difficult to predict the onset of a heating with any confidence.

Bulk coal self-heating tests have been limited due to the expense and time taken to obtain results. Some success has been obtained with various column-testing arrangements (Li and Skinner, 1986; Stott and Chen, 1992; Akgun and Arisoy, 1994; Arief, 1997), but the equipment used has not gained wide acceptance.

A new laboratory has been established within the School of Engineering at The University of Queensland (UQ) that uses a two-metre column to conduct a practical test capable of providing reliable data on coal self-heating (Beamish et al., 2002). This can be used to predict the onset of coal self-heating with acceptable engineering certainty for risk management purposes. Preliminary results from this new work are providing definitive insights into hot spot development (Beamish and Daly, 2004; Beamish, in press). This paper presents some examples, which show the effects of gas and moisture removal from coal.

COLUMN SELF-HEATING

Equipment
Beamish et al. (2002) describe the basic operation of the UQ two-metre column, which has a 62 L capacity, equating to 40 - 70 kg of coal depending upon the packing density used. The coal self-heating is monitored using eight evenly spaced thermocouples along the length of the column that are inserted into the centre of the coal at each location (Figure 1). Eight independent heaters correspond to each of these thermocouples and are set to switch off at 0.5°C below the coal temperature at each location so that heat losses are minimised and semi-adiabatic conditions are maintained radially.

Sample preparation
A fresh run-of-mine high volatile bituminous coal sample was obtained from a Newcastle Region longwall mine for testing in the UQ two-metre column. The coal particle size was kept below 150 mm and a size distribution of the sample was determined prior to loading into the column. The average particle size of the coal was 8.19 mm, based on the procedure described by Kunii and Levenspiel (1991) for estimating the surface-volume average particle size from the size distribution of the coal. Three subsamples were taken at this stage to obtain data on the as-received moisture of the coal, which was determined to be 3.1 per cent.

Test procedure
A standard test procedure has been developed for UQ two-metre column coal self-heating tests. The coal was loaded into the column with three 20 L plastic buckets. Once all the coal was in the column it was sealed and the heaters used to set the starting coal temperature, which in this case was 30°C. This was achieved overnight. Air was then introduced to the coal at 0.25 L/min. A computer records all data at ten-minute increments. The column has several safety devices including computer-controlled trips on the external heaters and a temperature trip on the air inlet line. These were set to ensure maximum safety during operation of the column.

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Following the successful completion of this test, which developed a 220°C hot spot, the coal was allowed to cool and equilibrate at 30°C. In this state, the coal in the column resembles a gas-drained situation, as both seam gas and moisture have been removed from the coal. An airflow of 0.25 L/min was again applied to the column.

As each column test progressed regular gasbag samples were collected from the outlet. These were analysed by Simtars using standard gas chromatography.

RESULTS OF COLUMN TESTING

Hot spot development in gassy, as-mined coal at 30°C

Figure 2 shows the temperature profile changes that take place with time in the column. In the initial stages of self-heating, a warm spot appears at 127 cm from the air inlet and progresses slightly downwind as the coal temperature continues to rise in this region. At around 85°C, the rise in the coal temperature in this region begins to slow and plateaus just below 100°C. Meanwhile, coal closer to the air inlet has been continually drying out and shortly after day eight a definite hot spot appears at 73 cm from the air inlet. Closer inspection of Figure 2 reveals that visible signs of the hot spot forming lower in the column begin as early as day six. The hot spot then begins to migrate towards the air source as the coal on the leading edge of the hot spot dries out and the hot spot chases the air to sustain the oxidation reaction. By day 11 a large hot spot in excess of 220°C is present 55 cm from the air inlet. At this stage the hot spot continues to migrate upwind as the coal dries out and also expands downwind due to convection.

Hot spot development in gas-drained, dry coal at 30°C

The methane content of the outlet gas for this test at 30°C was 0.8 per cent, compared with 2.1 per cent for the previous test at the same temperature. Similarly, the coal moisture content had reduced to less than 2.5 per cent (from a simple mass balance check and coal sample checks). Hence, there had been significant methane desorption and moisture losses during the hot spot development of the initial test, particularly in the region of highest coal temperature.

Hot spot development in the column for the non-gassy, dry coal test is summarised in Figure 3. While some warming appears in the upper half of the column, it is immediately overshadowed by a rapidly forming hot spot 55 cm from the air inlet (Figure 3). In this state there is no stepped rise in maximum coal temperature as the coal nearest the inlet is already predisposed to allow the air to reach oxidation sites in the coal. In other words the inhibiting effects of the seam gas and coal moisture in the pores of the coal have been removed.

Gas evolution in response to coal oxidation and hot spot development

The coal sample contained seam gas rich in methane with a subordinate amount of carbon dioxide and traces of ethane. Consequently, there are some noticeable differences between the gas evolution patterns of the gassy, as-mined and non-gassy, dried coal self-heating tests. Methane evolution is shown in Figure 4. The initial gasbag for both tests shows elevated levels of methane due to the gas desorbing in a static air environment during temperature equilibration to 30°C. In the as-mined coal, there is a significant rise in the methane concentration due to gas desorption up until the coal reaches a temperature of 83°C. Beyond this temperature the methane concentration drops rapidly. No such feature is evident for the drier coal.

Appreciable quantities of hydrogen are evolved as the coal self-heats from 30°C to 120°C (Figure 5). This pattern is present for both the gassy and non-gassy coal and occurs in similar amounts. Consequently, the hydrogen is not being produced from seam gas desorption. At higher temperatures the hydrogen evolution appears to level off. Grossman, Davidi and Cohen (1993) and Nehemia, Davidi and Cohen (1999) have reported research into the mechanisms responsible for hydrogen production from low temperature coal oxidation. Their findings will be discussed later in this paper.

Ethane evolution shows a prolonged increase in concentration (Figure 6), which unlike methane extends beyond 83°C. The non-gassy coal shows a roughly linear increase in ethane concentration from 60°C onwards. This tends to suggest that ethane is being produced predominantly from low temperature coal oxidation of the non-gassy coal. However, in the case of the gassy coal ethane from both gas desorption and oxidation mechanisms appears to coexist.
Ethylene is not evolved in detectable quantities until the coal temperature exceeds 120°C (Figure 7). This temperature appears to be consistent for both gassy and non-gassy coals. The increase in ethylene concentration beyond this temperature is reasonably linear. There is a shift in the ethylene evolution trend above 180°C for the gassy coal, which also corresponds to a drop in ethane evolution. This tends to suggest these two gases are closely linked in terms of their evolution mechanism.

**Gas indicator ratios in response to coal oxidation and hot spot development**

One of the key gas indicators used by the coal industry is Graham’s ratio. This is calculated as follows:

\[
GR (\%) = \frac{(CO \times 100)}{\text{oxygen deficiency}}
\]

where:

\[
\text{Oxygen deficiency} = \frac{(20.93/78.11 \times N_2 - O_2)}{78.11/20.93 \times (N_2 - O_2) - \frac{0.035}{78.11} \times N_2}
\]

It should be noted that the factor for the nitrogen calculation is different to values quoted in most reference texts as the value used in these is for nitrogen plus inerts. However, the gas analyses provided by Simtars give the nitrogen as a separate value.

There is a noticeable difference between the Graham’s ratio values of the gassy coal compared with the non-gassy coal (Figure 8). The non-gassy coal has a much higher Graham’s ratio for a given temperature, presumably due to the ease of air access to oxidation sites. This finding has consequences for interpreting gas monitoring results in areas of a mine that has been gas-drained.

Young’s ratio is another key indicator of coal self-heating progression. This ratio is calculated as follows:

\[
YR = \frac{CO_2}{\text{oxygen deficiency}}
\]

where:

\[
\text{Oxygen deficiency} = \frac{(20.93/78.11 \times N_2 - O_2)}{78.11/20.93 \times (N_2 - O_2) - \frac{0.035}{78.11} \times N_2}
\]

\[
CO_2 = CO_2 - 0.035/78.11 \times N_2
\]
**Figure 5** - Hydrogen evolution as a function of maximum coal temperature.

**Figure 6** - Ethane evolution as a function of maximum coal temperature.

**Figure 7** - Ethylene evolution as a function of maximum coal temperature.
At temperatures less than 90°C, the Young's ratio of the gassy coal is higher than for the non-gassy coal (Figure 9). The difference appears to be too great to be explained by the presence of additional carbon dioxide in the seam gas, which is present in only minor concentrations. Therefore, the additional carbon dioxide must be produced by another mechanism. At temperatures higher than 90°C, the Young’s ratio of the non-gassy coal is higher than for the gassy coal, which is the equivalent trend to the Graham’s ratio.

The H2/CO ratio shows exactly the same trend for both gassy and non-gassy coal (Figure 10). This ratio appears to reach a maximum near 80°C. The gassy coal reaches a higher maximum, which is almost three times that of the non-gassy coal, primarily due to the greater amount of carbon monoxide generated by the non-gassy coal. From this ratio it is clear that at temperatures lower than 80°C, the bulk coal self-heating from low-temperature oxidation of the coal favours a hydrogen generating mechanism. Once the coal temperature exceeds this value the carbon monoxide generating mechanism is favoured.

Nehemia, Davidi and Cohen (1999) have suggested that the low-temperature oxidation mechanism for the hydrogen production is decomposition of formaldehyde catalysed by coal:

\[
2\text{CH}_2\text{O} + \text{O}_2 \xrightarrow{\text{coal}} 2\text{H}_2 + 2\text{CO}_2
\]

It is interesting to note that the by-product of this reaction is carbon dioxide. The imbalance shown by the Young’s ratio for the gassy coal may be strong evidence to support this reaction mechanism. Chamberlain, Barrass and Thirlaway (1976) also noted reasonable quantities of aldehydes being evolved from coal oxidation in this temperature range, which would further support this mechanism as a likely source of the hydrogen.

**HOT SPOT DEVELOPMENT FEATURES OF GASY AND NON-GASSY COAL**

The hot spot development features seen in the UQ two-metre column test are entirely consistent with the moist coal self-heating models of Schmal, Duyzer and van Heuven (1985), Arisoy and Akgun (1994), Portola (1996) and Monazam, Shadle and Shamsi (1998). In particular, the moist coal model of Schmal, Duyzer and van Heuven (1985) predicts the plateau effect of the initial hot spot development (Figure 11). They maintain heat effects due to evaporation and condensation of moisture is responsible for the coal reaching a constant maximum temperature of 80 - 90°C. This level continues until the coal becomes dry locally, after which a steep temperature rise occurs at the dried spot.
In all these models, it is clearly shown that the higher the moisture content of the coal, the longer it takes to reach dangerous temperatures. The difference between gas-drained, dried coal and the as-mined, moist coal illustrates this feature (Figure 11). In fact, the maximum temperature curve for the gas-drained, dry coal is a direct match of the dry model proposed by Schmal, Duyzer and van Heuven (1985). It took 4.25 days to reach temperatures in excess of 150°C for the dried coal condition and 8.5 days to reach the same stage in the moist coal. Equally important, is the fact that for the dried coal, the migration towards the air source is much faster as heat is not used up to evaporate moisture from the coal on the leading edge of the hot spot.

These findings have major implications for coal mines practising coal seam gas drainage. It is well known that gas-drained coal is dry and dusty when mined. Consequently, any Spontaneous Combustion Management Plan must consider the elevated risk of coal self-heating that results from gas drainage. Small-scale R20 testing by Beamish, Barakat and St George (2001), which is a dry coal test, shows that as rank decreases below medium volatile bituminous, the self-heating rate of the coal increases dramatically. Hence, gas drainage of coals in the low rank high volatile bituminous range will create a higher risk of self-heating than for the non-drained coal in the same mine.

To mitigate the elevated risk of self-heating from gas drainage there is a need to consider returning moisture to the coal. This could be achieved through water infusion. The difficulty encountered here is how efficient are the procedures for doing this. Future research into this area would be most beneficial to all underground coal mines, as the water infusion would also help with dust suppression during mining.

CONCLUSIONS

The UQ two-metre column is producing coal self-heating results that are consistent with both theory and practice. In particular the hot spot development matches closely with several published models for coal self-heating. Features of moisture transfer and hot spot migration are clearly visible in the column. Under the conditions used for testing a high volatile bituminous coal, a hot spot reached thermal runaway (>150°C) after 8.5 days from a gassy, as-mined state. The same coal in a gas-drained, dried state reached thermal runaway after only 4.25 days. These results indicate that it is sensible to consider some form of water infusion for coal that has been gas-drained.
There are significant differences in the gas evolution patterns of gassy, as-mined coal compared with non-gassy, dried coal. Removal of seam gas and moisture allows easier access of air to oxidation sites, with a resultant higher Graham’s ratio for any given coal temperature. However, an increasing Graham’s ratio is still a good indicator of coal self-heating in both cases. Substantial quantities of hydrogen are evolved at low temperatures during bulk coal self-heating. Hence, the old adage of hydrogen acting as an indicator of an advanced heating in its own right is a fallacy. The evolution of ethane from gassy coal appears to be a mix of gas desorption and low temperature oxidation, with both mechanisms responding to temperature increase. Measurable ethylene does not appear until the coal temperature has reached 120°C in both gassy and non-gassy coal, and continues to increase as the coal temperature increases beyond this value.

Further column testing is in progress on a range of Australian coals to look at hot spot development features in more detail and to provide the coal industry with a better means of assessing the risk of coal self-heating.

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Spontaneous Combustion in Open Cut Coal Mines — Recent Australian Research

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ABSTRACT

The control of spontaneous combustion in spoil piles is an area of concern for a number of open cut coal mines. Spontaneous combustion in spoil may occur when carbonaceous waste materials are exposed to air. In large piles, the relatively high voidage within the pile may allow sufficient movement of air through the pile to sustain oxidation and heat generation.

Over the past decade, ACARP and CSIRO have funded a number of projects aimed at providing a better understanding of the causes of spontaneous combustion in spoil piles and the development of control strategies. Work has focused on prevention measures but has also considered measures available to deal with well-developed fires. Field, laboratory and numerical modelling methods have been brought to bear on the issues.

Individual projects have addressed:

- causes of spontaneous combustion including the relative reactivities of mine waste materials;
- the use of cover layers to prevent spontaneous combustion;
- the use of flyash grout to control an active fire;
- the emissions of air pollutants; and
- the emission of greenhouse gases.

This paper provides a description of the work carried out to date with an emphasis on the major findings of the research and its application to open cut mines.

CONTROL AND PREVENTION OF SPONTANEOUS COMBUSTION IN SPOIL PILES

Spontaneous combustion in open cut mines

Self-heating and spontaneous combustion in coal mining spoil piles is a topic that has received rigorous scientific consideration only relatively recently. The main drivers have been:

1. The persistence of neglected old spoil fires, requiring that fire control measures be applied. While there have been many different techniques developed in Europe, USA, China, India and Thailand, no single technique provides assured success.

2. The scale of large open cut operations, where the environmental impact of spoil pile fires can be significant, requiring spontaneous combustion prevention and control measures.

A recent review by Walker (1999) has considered the major aspects of fires in coal mines, due to spontaneous combustion. This work has been practical in approach and has built on previous more fundamental approaches as reviewed by Glasser and Bradshaw (1990) and Carras and Young (1994).

In the USA, the Department of the Interior, Office of Surface Mining Reclamation and Enforcement maintains a database of 150 uncontrolled abandoned coal mine fires, which includes 80 surface fires with an estimated reclamation cost of US$ 300 million. There has never been a comparable study of uncontrolled coal fires in Australia. Most Australian sources refer only to reportable spontaneous combustion incidents in operating mines.

Various techniques for extinguishing uncontrolled fires in abandoned mines and spoil piles have been tried by the US Bureau of Mines (Kim and Chaiken, 1993). These include:

- excavation,
- flooding and quenching,
- bulk filling,
- surface sealing,
- inert gas injection,
- chemical treatment, and
- burnout control.

The relative effectiveness of the various methods of fire control can range from high to poor. According to a review carried out in 1992 by the USBM, excavation, creating barriers and sealing generally rank from high to low in probability of success. Cost is also a major factor, with systems that involve extensive excavation incurring the highest costs.

For spontaneous combustion prevention, precautionary measures such as encapsulation of reactive material and compaction to deny oxygen access, are generally recognised as the best way of minimising the risk for spontaneous combustion becoming a problem in the future. However, no quantitative guidance is available as to the minimum thickness of cover layers or the minimum voidage to ensure successful prevention.

Aspelung and Adamski (2002) provided a report on the spontaneous combustion prevention practice at the Grootegeeluk mine in South Africa, which has been in operation since 1980. Until 2001, all wash-plant discard and carbonaceous interburden materials were stacked on out-of-pit dumps and spontaneous combustion of the waste material resulted on the discards dumps. During the period 1980 - 1988, many large-scale and laboratory tests had been conducted to determine the factors that contribute to spontaneous combustion, but no successful method of preventing or containing the problem had been found. After large-scale tests in 2000 - 2002, in-pit backfilling of the reactive waste materials into pre-built inert compartments was identified as the preferred method for prevention of spontaneous combustion, based on spontaneous combustion risk, safety and cost. The practice now adopted in the ten-year mine plan at Grootegeeluk, is to backfill in three tiers of compartments, each 30 - 35 m high/ deep, to a final height of 143 m. This includes two intermediate cover layers of weathered overburden, each 3 m thick, and a final cover layer of 9 m thickness and a topsoil layer of about 1 m. The compartments have widths of 130 - 300 m, and intermediate dividing walls, either pre-placed or tipped at the batters, composed of a minimum of 5 m of overburden and 25 m of sandstone.

Australian research

ACARP has co-funded a number of research projects into the causes of spontaneous combustion and the development of appropriate control and prevention methods. These studies have been conducted by CSIRO and ACIRL, working in collaboration.

Carras et al (1994) measured the intrinsic oxidation rate of coal and carbonaceous shales routinely sent to spoil at five open cut coal mines in the Hunter Valley. The oxidation rate, which is directly linked to the spontaneous combustion propensity of the samples, was found to be a linear function of the non-mineral content.
carbon content of the sample. Measurements of the heat generated by oxidation of the carbonaceous shales was measured and shown to be similar to that produced from coal oxidation. This meant that essentially the same chemical processes were involved in each case.

Carras et al (1994) also installed temperature and oxygen concentration measurement probes at three mines in the Hunter Valley to provide fundamental information on self-heating and active combustion in large spoil piles. These measurements were continued by Carras et al. (1999) with field measurements in spoil piles having been made for periods up to 6.5 years. The probes provided unique data on the phenomenology of heating, and cooling spoil piles. In particular a hot spoil pile at one mine was battered and covered with a clay layer (5 - 15 m thickness) as a means of combating spontaneous combustion. Figure 1 shows the temperature measurements, for this spoil pile, as a function of time since the installation of the probes.

The data in Figure 1 has shown a temperature decline, at the hottest location (19 m) from ~260°C to 65°C during the 6.5 years over which measurements were made.

In a subsequent project, Carras et al (1997) extended the CSIRO mathematical model of self-heating (SPLGOF) to predict self-heating in spoil piles. In addition, two small spoil dumps were constructed at a Hunter Valley mine and instrumented with temperature and oxygen probes. The material comprising the dumps was subjected to laboratory oxidation rate measurements so that its reactivity could be quantified. One of the dumps underwent self-heating and provided the opportunity to compare directly the predictions of the CSIRO model of self-heating with measurements of the temperature and oxygen profiles. Allowing for the approximations necessary to run the model, the model predictions were in good agreement with the data from two of the probes but not the third, indicating that while considerable progress had been made in the modelling of self-heating there were still mechanisms which needed to be included.

Carras et al. (1997) also undertook a trial of flyash grouting as a fire control method on a section of hot spoil at one of the Hunter Valley mines. Both dry flyash and a water-flyash slurry were used as the grout. The trial took place in a section of spoil, which had been instrumented with four temperature and oxygen probes and contained an active fire. A total of 108 tonnes of flyash was injected over a period of three days. While the grouting had an immediate effect on the temperatures in the spoil pile it is now apparent that grouting was successful in extinguishing the spoil pile fire in a very small volume of the order of 100 m³.

In 1997 dollars, the cost of the grouting trial was $18 000. It is considered that this cost could be reduced significantly for grouting carried out in a large-scale commercial project. Nevertheless the relative cost of the grouting compared with other options, such as excavation, water quenching or covering is probably prohibitive for all but the most critical applications.

An important outcome from this series of projects was the guide on spoil pile self-heating management prevention and control produced by Haneman and Roberts (1997). The guide was based upon the experience of four Hunter Valley coal mines and incorporated the research findings of Carras et al. (1994, 1997). The guiding principles were:

- identify and control all fuel sources going to spoil,
- minimise the quantity of fuel going to spoil,
- reduce oxygen pathways in spoil piles, and
- avoid dumping hot materials.

Two small spoil dumps were subjected to laboratory oxidation rate measurements for the approximations necessary to run the model, the model predictions were in good agreement with the data from two of the probes, but not the third, indicating that while considerable progress had been made in the modelling of self-heating there were still mechanisms which needed to be included.

Carras et al. (1997) also undertook a preliminary investigation into the properties affecting the performance of cover layers in inhibiting oxygen penetration into spoil. Measurements of the diffusion rate of oxygen through three inert overburden materials, including a clay from one mine, pre-strip material from another mine and marine conglomerate from a third mine, showed that the diffusion coefficients for oxygen depended on the air filled voidage of the spoil. Calculations of the penetration of oxygen by diffusion through an inert surface layer show that very thick layers of low voidage material are required to significantly reduce the flux of oxygen. Such low voidages are not normally achieved by tipping and compaction of dry spoil. Water is required in the surface layer to reduce the diffusive flux of oxygen significantly. Consequently, the water holding properties of potential cover materials are critical.

In the most recent work, Roberts et al (2004) extended the research to model directly water transport in the cover layers and the self-heating properties of the spoil. Mathematical modelling was carried out by using the two computer codes SPLGOF (Carras et al., 1997) and SWIM (Verburg et al., 1996). SPLGOF solves the equations of heat and mass transfer for a porous body and accounts specifically for oxygen transport, reaction with the carbonaceous material and release and transport of heat. Input to SPLGOF includes the air-filled voidage of the cover layer and of the spoil, as well as the reactivity of the spoil. The latter depends on the carbonaceous content of the spoil and its particle size distribution. SPLGOF only considers heat and mass transfer by the transport mechanism of diffusion.

SWIM (also developed by CSIRO) calculates the transport of water within soils and was used to simulate the infiltration of rain and the flow of water through the cover and spoil. The presence of water acts to reduce the air-filled voidage of the cover layer
and hence the ingress of oxygen and the subsequent rate of self-heating. As SWIM does not include thermal effects, the two models SPLGOF and SWIM were coupled in order to account for water flow, evaporation and spoil self-heating.

Computer simulations were performed using historical Hunter Valley, NSW, weather records and a range of cover and spoil properties. The results of the computer simulation results showed very different degrees of self-heating depending on the initial conditions. For instance, if the cover layer was initially wet, with the lower spoil reactivities, the simulated spoil piles showed very little self-heating even after 24 years. If the covers were applied dry over the more reactive spoil, the simulations predicted run-away heating in some cases but not others, depending on the nature of the cover material and the thickness of the cover.

The mathematical modelling also showed that the ingress of oxygen into the cover depended strongly on the occurrence of rain events. The oxygen flux dropped markedly with infiltration into the cover following rain, but returned to pre-rainfall, near-steady values during dry periods.

In order to obtain an empirical ranking of cover materials, and as a further method to provide a test of the computer model predictions, an area of spoil pile at a Hunter Valley mine was used to create eight cover layer plots from six different materials. An oxygen fluxmeter was used to measure the oxygen flux into the spoil both prior to and after a period of six months after the covers had been emplaced (Timms, 2002). The values of oxygen flux into the spoil showed wide variation reflecting the high variability of spontaneous combustion activity and of the properties of the near surface spoil. When the average of all the oxygen flux values for bare spoil was compared with the average values of oxygen flux through the covers, a reduction in the flux of oxygen into seven of the eight cover plots was observed. However this result is sensitive to the value of oxygen flux attributed to bare spoil, and given the large variability of the latter, this result cannot be taken as conclusive. However, the magnitude of the measured oxygen fluxes through the different covers, and their ranking, were in broad agreement with the computer model predictions.

The field and laboratory work showed that the most significant properties of a cover layer material were the water retention characteristic and saturated hydraulic conductivity of the unconsolidated (‘minesoil’) material. These hydraulic properties were determined for typical overburden strata at two Hunter Valley mines, by size analysis and using published correlations to relate the size distribution to the water retention function and hydraulic conductivity.

The main conclusions from the field-work and the computer model simulations, were the following:

1. the use of a cover layer can significantly reduce the rate of self-heating of a spoil pile;
2. the three materials used in the simulations, namely clay, marine conglomerate and weathered sandstone, can be ranked in that order as to their effectiveness as cover layer materials, according to their soil-like hydraulic properties, ie the water retention characteristic and saturated hydraulic conductivity;
3. the computer modelling showed that:
   • cover effectiveness depends on the cover thickness with thicker covers being more effective than thin ones;
   • cover thicknesses (1 - 2 m) of highly water-retentive clay-rich materials limited self-heating in spoil, for the spoil reactivities assumed and under Hunter Valley conditions; and
   • the use of sufficiently thick (greater than 5 - 10 m) layers of the low water-retentive materials, such as sandstone, can also reduce the likelihood of self-heating.

A practical methodology for predicting cover layer effectiveness was developed from the computer simulations. From fundamental considerations, the relative effectiveness of a cover layer will depend on:

1. the composition, particle size distribution and bulk density of the cover layer;
2. the water content of the cover layer;
3. the air filled voidage of the cover layer, which in turn depends on points 1 and 2 above;
4. the heat transport property (thermal diffusivity) of the cover layer, which in turn depends on points 1 to 3 above; and
5. the oxygen transport property (oxygen diffusivity) through the cover layer, which in turn depends on points 1 to 3 above.

The relative effectiveness of a cover material can be developed by considering the ratio of its thermal diffusivity to its oxygen diffusivity. The manner by which this can be related to soil properties are described in detail by Roberts et al (2004).

Roberts et al (2004) also develop a stability index (S) for a cover material with effectiveness (C) and thickness (L), for a particular location (ie climatic conditions). The stability index can be expressed as:

\[ S = C/L \]

The stability index was determined from the computer simulations and is a measure of whether a spoil pile of a certain reactivity, cover layer effectiveness and thickness will lead to spontaneous combustion, or not. The above expression shows that greater cover effectiveness (ie high thermal diffusivities and low oxygen diffusivities) and greater thicknesses, lead to higher stability, while the inverse is true for less effective cover materials and thinner layers.

The results of the computer simulations from Roberts et al (2004) based on real mine materials are shown in Figure 2, where the cover material effectiveness has been plotted as a function of bulk density for different sand contents of the cover layer material. Clays through to sands are covered. As anticipated the clay like materials are clearly most effective. While the parameters of the computer simulations were matched, as far as possible, to Hunter Valley conditions, there are a number of issues which require further consideration before the results shown in Figure 2 can be put to routine operational use (see Roberts et al, 2004). Briefly, they include:

- The role of vegetation and its effect on the subsurface water profile.
- The modelling assumption of uniformity of properties within particular layers.
- The reactivity data of the spoil used in the current work were obtained by extrapolation of laboratory measurements. In situ values should be obtained and used.
- The representativeness of the hydraulic and thermal properties of cover materials and spoil.
- The impact of run-off on slopes, non-uniform surfaces, and large contiguous voids in spoil, which may lead to channelling of water and air rather than the diffusional transport processes, which were assumed in the modelling.

GREENHOUSE GAS (GHG) EMISSIONS FROM SPONTANEOUS COMBUSTION

One of the environmental issues of increasing significance for spontaneous combustion is the emission of greenhouse gases. While spontaneous combustion of coal has been recognised by the Inter-Governmental Panel for Climate Change (IPCC) as a...
potential source of greenhouse gas emissions, it has been excluded from greenhouse gas inventories as it is considered that there is no acceptable method for estimating the emissions.

In recognition of this, ACARP and CSIRO have carried out three projects to explore methods for establishing greenhouse gas emissions from spontaneous combustion. The first (Carras et al., 2000) sought to provide methods, supported by direct measurement, to quantify the emissions of greenhouse gases. Measurements of emissions from spoil piles, coal rejects and tailings were conducted at 11 mines in the Hunter Valley in NSW and the Bowen Basin in Queensland using a chamber technique. The results of this work led to the development of emission factors for several broad categories based on the extent of spontaneous combustion present. These were:

- Category 1 – intense spontaneous combustion characterised by smoke and steam, major cracks, surface discolouration and obvious signs of venting.
- Category 2 – spontaneous combustion with less well pronounced signs, small cracks, surface discolouration and occasional wisps of smoke and steam.
- Category 3 – no sign of spontaneous combustion.

While the project provided the first systematic study of greenhouse gas emissions from spontaneous combustion in open cut coal mines, there were practical problems in applying the results to estimate greenhouse gas emissions from operating mines. While the chamber technique provided direct emission measurement, it was labour intensive and required many measurements to obtain representative values. In addition, it is difficult to obtain an objective measure of the extent of spontaneous combustion in spoil piles. However, one of the key findings of the research was that the emissions from spoil piles without spontaneous combustion were similar to the background emissions from vegetation and biota from surfaces such as forest floor and domestic lawns. This suggests that the carbonaceous material within the spoil piles is not being exposed to oxygen and hence not contributing in a significant way to greenhouse gas production.

Due to the difficulties with the chamber approach Carras et al. (2002b) investigated the use of remote sensing techniques such as airborne infra-red thermography to investigate whether more accurate and cost-effective monitoring of the extent of spontaneous combustion in spoil piles and the associated greenhouse gas emissions could be achieved. This approach was based on the finding by Carras et al. (2000) that an approximate relationship existed between the emission rate of greenhouse gases and the average surface temperature of the spoil pile surface. Figure 3 shows a plot of the cumulative area greater than a given temperature for the same section of spoil over a 13 month period and for two different pixel resolutions i.e. $7 \times 7$ m and $2 \times 2$ m.

The results in Figure 3 show that for the $7 \times 7$ m pixels, the total area greater than a given temperature has changed over the 13 month period. The data from the $2 \times 2$ m pixels shows greater areas at the higher temperatures due to the enhanced resolution’s ability to capture local ‘hot spots’. However, the above results are subject to a number of assumptions in their analysis and require detailed ground validation.

From data such as those shown in Figure 3, Carras et al. (2002b) concluded that airborne infra-red data could be used to monitor the long-term behaviour of spoil piles subject to spontaneous combustion. The same data could also be used to estimate greenhouse gas emissions for spontaneous combustion. However, due to the complexity of the processes involved in producing heating and its surface manifestation and the associated emissions of greenhouse gases, the emissions estimates were still subject to significant uncertainty. Nevertheless (and the scatter in the data notwithstanding) over a suitable time period the method could be used to monitor the progress of spontaneous combustion in spoil piles and associated emission of greenhouse gases.
SPONTANEOUS COMBUSTION IN OPEN CUT COAL MINES — RECENT AUSTRALIAN RESEARCH

A further approach which has been considered in estimating emissions from spontaneous combustion requires the direct measurement of emission fluxes by mapping out the concentration of GHG as a function of crosswind distance, height above the ground and wind speed, to allow a direct estimate of the emission rate. This technique has been employed previously by Carras et al (1991) and Carras et al (2002a) to measure:

- the fluxes of methane emitted from Sydney, Melbourne and Brisbane;
- the volatile organic compounds (VOC) emitted from the Kwinana industrial region; and
- the emissions of VOC and nitrogen oxides (NOx) emitted from Hong Kong.

While this approach is the most direct it requires traverses of the downwind plume at various heights above the ground. However, ground level sources require very low level passes (<50 m) through the plume. While these can be achieved above the sea such traverses are vastly more difficult and dangerous above land. Thus and notwithstanding the appeal of this approach for open cut coal mines, safety issues preclude this option.

A variation of the direct flux measurement approach described above is to traverse a ground based CO2 detector across the plume and to use knowledge of micrometeorology and plume dispersion to estimate the horizontal and vertical extent of the plume and hence calculate the emission rate. This approach was used by Williams et al (1993) in their estimates of methane fluxes from open cut coal mine.

In addition to the methods described above to determine area source emissions, another approach that has been gaining widespread use in recent years, particularly with regard to global greenhouse gas emissions, is the use of inverse modelling methods. Inverse methods, as the name implies arise from inverting the normal advection diffusion equation used to describe transport of gases and particles in the atmosphere. For instance, in conventional applications of atmospheric transport models such as air quality models, source strength and meteorological data are used as inputs to calculate the concentration of a species at downwind locations. By contrast, an inverse technique involves back-calculating the strength of an emissions source using a measured concentration time series at one or a number of selected sites.

While there has been considerable work on methods to invert air pollution concentrations in order to obtain estimates of the emission strengths of sources, developing a completely general inversion method is not possible as information is lost as the pollution cloud is transported and diffuses in the atmosphere. Other physical information is required to constrain the possible solutions to the equations and yield realistic results. In general, the greater the number of monitors and the ‘cleaner’ the signal from an individual source the better the solution to the inverse problem. In addition and for the case of CO2 emissions from spontaneous combustion in the Hunter Valley, the importance of the concentrations from other sources in the Valley such as power stations and the major highway must be taken into account in order to design an appropriate experimental approach to provide greenhouse gas emissions from spontaneous combustion.

Lilley and Carras (2003) modelled the large sources of CO2 in the Hunter Valley using a computer-based air quality model (TAPM, Hurley, 1998), which has been widely used in air pollution studies in Australia. An investigation of CO2 sources in the Upper Hunter Valley showed that spontaneous combustion and power stations can give rise to significant concentrations at ground level. However the impact of the power stations emissions are most pronounced during the day time hours while the impact of the spontaneous combustion emissions are most pronounced during the night time. This is because the former are elevated while the latter are ground level sources. This suggests that concentration measurements should focus on data during the night time period. While the emissions from road traffic and rail are significant, their ground level concentration signature is not as pronounced as for the other two major sources.

Consideration of results of the air quality modelling suggests that monitoring sites for the inverse modelling should be sited such that:

- the sites are sufficiently close to the spontaneous combustion sources to enable a large measurable signal;
- the sites are chosen on the basis of meteorology to best capture the likely CO2 spontaneous combustion signal; and
- the sites are chosen to minimise the influence of other sources.

The above work is continuing through a current ACARP project, which is applying the methodology developed and is due to be completed by mid 2006.

AIR POLLUTION

In addition to greenhouse gases, other air pollutants are emitted from spontaneous combustion. These include:

- SO2 (arising from sulfur associated with coal, either as mineral matter or bound to the organic fraction);
- NOx;
- CO due to incomplete combustion;
- fine particles;
- other hazardous air pollutants (eg Polynuclear aromatic hydrocarbons, PAH).

There has been relatively little work on these emissions that is available in the open literature. Carras et al (1999) carried out a study at a Hunter Valley mine where the exposure of workers, within the cabin of a bulldozer, working in the vicinity of spontaneous combustion was measured. These results showed that the PAHs measured within the cabin were below the values expressed by the occupational health and safety guidelines. In outside air and in close proximity to spontaneous combustion...
fires, the PAH levels may be sufficiently high to result in exposures which may be greater than the recommended values. Further work is required to quantify the emissions of these pollutants and to assess their overall significance.

CONCLUSION
The work carried out through the ACARP projects described above has advanced considerably the fundamental understanding of the processes responsible for spontaneous combustion. The research has also led to practical guidance that can be used in an operational manner at operating open cut coal mines.

The outstanding issues for further research, remain:
1. field validation of the predictions of the mathematical models for preventing self-heating;
2. development of a method of adequate accuracy that can be applied in a routine manner to determine greenhouse gas emissions from spontaneous combustion; and
3. an assessment of the overall emissions from spontaneous combustion.

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3. an assessment of the overall emissions from spontaneous combustion.

REFERENCES


Results of Self-Heating Tests of Australian Coals Conducted in a 16 m³ Reactor

F Clarkson¹

ABSTRACT
A research and testing facility has been developed at Simtars to allow the spontaneous combustion testing of coal on a large scale. In this adiabatic reactor, the coal particle sizing is closer to that which could be found in the goaf and therefore the reaction is more likely to simulate the underground situation. This allows aspects of the spontaneous combustion process and methods for detecting a heating to be investigated which cannot be done in an operational mine. Up to 16 tonnes of run of mine or crushed coal is loaded into the reactor and allowed to self-heat while the temperature and gas profiles within the coal pile are monitored. The gases evolved from the coal are analysed using a fixed gas monitor system and gas chromatography to determine which of the common mine fire indicators and ratios can be used to predict the onset and progress of coal heatings. Benchmarking of the inherent propensity of a coal to spontaneously combust at both the large- and laboratory-scale is also possible by comparing the sample against previously tested samples.

It has been found that the coal particle size affects both the time it takes for the coal to react and the location of the developing heating. The heatings generate approximately one metre from the inlet but in the crushed coal case do not progress to an advanced heating stage until the ‘hot spot’ migrates towards the coal surface where there is an increased oxygen supply. Physical properties such as the compressive strength of the coal also affect the heating size and the time it takes for a heating to develop. Higher strength coals appear to oxidise more slowly and therefore take longer to self-heat as oxidation of the coal is the heat-generating step in the spontaneous combustion process.

The best indicators of the state of the heating were found to be CO make and Graham’s ratio which are independent of air flow. Investigation of a potential spontaneous combustion should begin as soon as the CO make or Graham’s ratio exceed the background levels. The results from the tests undertaken on a number of Queensland and NSW coals are reported here together with the implications for the mining industry.

INTRODUCTION
Large-scale testing of coals aimed at gaining a better understanding of the processes involved in spontaneous combustion have been undertaken around the world for over a hundred years. The mass of coal used in recent tests has ranged from one to five tonnes of dry ground coal (Chauvin, Lodel and Philippe, 1985) in the Centre d’Etudes et Recherches de Charbonnages de France (CERCHAR) tests, ten tonnes in the Safety in Mines Research Station at Buxton in the UK (Mason and Tideswell, 1939), to 13 short tons of coal in the United States Bureau of Mines (USBM) tests (Smith, Miron and Lazzara, 1991). These reactors have been used to study the heating profiles of various coals enabling assessment of the risk of spontaneous combustion in a mine, during transportation, in stockpiles and development of mathematical models in the case of the CERCHAR tests. The detection of heatings by smell and by monitoring of carbon monoxide levels were investigated in the Safety in Mines Research Station tests at Buxton. The dependence of self-heating on coal reactivity, particle size, freshness of the coal surface, heat of wetting, and availability of oxygen were studied in the USBM tests.

The 16 m³ adiabatic reactor Simtars has been developing for the last eight years is similar to the USBM reactor and requires 16 to 18 tonne of coal depending on the particle size of the test coal. Early tests in the Simtars reactor involved three coals from mines with known histories of spontaneous combustion in Queensland and New South Wales. These tests showed that although initial self-heatings were achievable, they did not proceed to thermal runaway due to heat losses from the reactor. Successful heatings were however obtained when the coal was artificially stimulated using a buried heating element.

Following installation of an insulated cover, a further five tests were undertaken on three New South Wales and two Queensland coals in order to investigate the effect of particle size, oxidation rate and initial coal temperature on the development of a heating. Analysis of the exhaust gas stream and also gases from selected sites within the coal pile was conducted to investigate the effectiveness of mine gas ratios currently used in Australia to detect a spontaneous combustion event.

EXPERIMENTAL
The apparatus described here is the current unit, which has been developed from the original reactor that had no thermal cover and only nine gas sampling ports. Details of the original reactor design were included in the final report for ACARP Project C5031 (Cliff et al, 2000b). Approximately 15 - 18 tonnes of run of mine or crushed coal were arranged in a pile 2 m wide, 2 m high and 4 m long between the two block walls of the reactor (Figure 1). One-metre long plenum chambers were established on both ends of the coal test section for uniform circulation of air through the coal. Two steel grids were used to separate the plenum chambers from the section of the reactor where coal was held. Air was passed from one end to the other of the sealed -reactor with a nominal internal air flow up to 100 L/min in order to allow the coal to dry out without excessive oxidation.

Gas analyses were conducted using a fixed O₂, CO, CO₂ and CH₄ monitor in addition to periodic analysis by gas chromatograph on samples taken from up to sixteen gas sampling tubes inserted into and across the central axis of the reactor and also the exhaust gas outlet (Figure 2).

Five layers each of 50 thermocouples were used to monitor the temperature profiles throughout the pile. The distance between any two thermocouples in the pile was 0.4 m. In addition the inlet and exhaust pipes of the plenum chambers are instrumented to determine the airflow and humidity. A data acquisition system was used to monitor both thermocouple readouts and the inlet and exhaust readouts for this project.

The current apparatus also includes a thermal cover to reduce the heat losses thus simulating a larger body of coal and facilitating the self-heating of the coal. The air space was heated using a recirculated air system, which included a blower and inline heater. Heating the air space formed meant that the equivalent ambient temperature could be controlled thus increasing the chances of a natural self-heating in the test coal. A second blower fed fresh air directly to the inlet plenum via a heat exchange box in the cover air space to ensure the inlet air temperature matched the temperature of the air space surrounding the coal.

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Figure 1 - Schematic of 16 m$^3$ reactor.

- **Fixed wall**
- **Coal pile exhaust**
- **Electrical and data acquisition boxes**
- **Airflow adjustment (for coal pile)**
- **Coal pile blower (supplies fresh air to coal pile)**
- **Recirculating blower (cover space)**
- **Insulated Cover roof - 200mm thick walls - 100mm thick**
- **Wheeled cover lift jacks (1 on each corner)**
- **Concrete Slab**
- **Concrete block construction**
- **Intake plenum door**
- **Intake duct with anemometer**
- **Floor distribution duct for recirculated heated air**
- **Insulated cover rolled back for access to Spon Comb Reactor**
- **Insulated cover over Spon Comb Reactor**
- **Heat exchange box for coal pile intake**
  (Note: Air return duct at top of cover void space is permanently fixed to the insulated cover)
- **Gas sampling spears within coal pile**
- **Screened sand layer above coal**
- **Exhaust plenum**
- **Mesh end panel supports coal pile**

25 longitudinal thermocouple ropes, each consisting of 10 thermocouples, 5 layers of 5 ropes equally throughout pile

**Note:** coal pile sectioned for clarity.
Results and Discussion

Tests without a thermal cover

One sample of Moura D and two samples of Dartbrook coal were allowed to self-heat for varying periods with airflows as outlined in Table 1. Full details of these coals and the test results were reported in the final report for ACARP Project C5031 (Chif et al., 2000b).

The initial heating of the Moura D sample was affected by heat losses due to the seasonal ambient air temperature, which could vary from 4 to 24°C during the day. Installation of a fan heater in the inlet plenum chamber stabilised the inlet air temperature allowing a maximum temperature of 56°C to be reached after 64 days. Further attempts to reduce the heat loss resulted in the peak pile temperature falling more than 20°C over a 12 day period. The self-heating phase was terminated at this point after 77 days and the buried heater element used to stimulate a heating.

The stimulated heating phase ran for a further 20 days while the temperature of the buried heater element was gradually increased from 80°C to 450°C over a nine day period in line with temperature increases in thermocouples up wind of the element. An intense smell of ‘fire stink’ was detected in the area close to the reactor at day 97, four days after the heating element was switched off. Infrared images of the exposed coal in the inlet plenum chamber showed the hot spot had migrated to the inlet coal face, reaching a maximum temperature of 488°C. The heating generated was finite in size decaying rapidly in temperature with distance from the inlet. The nearest thermocouple 200 mm downwind from the hotspot measured a temperature with distance from the inlet. The nearest thermocouple 200 mm downwind from the hotspot measured a temperature with distance from the inlet.

The Dartbrook 1 sample was allowed to self-heat for 51 days attaining a peak temperature of 63.9°C after 36 days. The self-heating phase of the test was terminated after 51 days when the peak pile temperature fell to 62°C. Experience gained from the previous tests had shown that a consistent drop in temperature was characteristic of heat losses to the surroundings exceeding the heat generated by the oxidation process. Further evidence to support this conclusion was obtained by monitoring the gases from the reactor exhaust, which indicated that the coal pile was in a steady state with no sign of self-heating occurring.

The stimulated heating phase of the test ran from days 51 to 85 of the test. The buried heater element was held at 400°C for 16 days and then increased to 500°C for a further five days before the buried heater was switched off. Infrared images on day 85 of the exposed coal in the inlet plenum chamber showed that a hot spot with a maximum temperature of 357°C had developed at the inlet coal face. As was found with the previous tests, the hot spot was relatively confined and inclined to move upwind to the exposed coal surface (Figure 3).

Analysis of the gas from the exhaust by gas chromatography (GC) found that no ethane or ethylene could be detected during the self-heating phase of these tests. Carbon dioxide was found to be present as seam gas and in the case of the Dartbrook samples methane was also present. Carbon monoxide levels when detectable were of the order of 5 - 15 ppm. Overall, the gas evolution data indicated that no exponential self-heating was occurring during the self-heating phase.

During the stimulated phase of the testing, hydrogen, ethane and ethylene were in general not detected in the reactor exhaust gas stream until the final day of each test when the hot spot reached the inlet coal face where the oxygen supply was unrestricted. Carbon monoxide and carbon dioxide levels showed significant increases during the last few days of the test which are characteristic of exponential thermal runaway.

<table>
<thead>
<tr>
<th>Coal</th>
<th>Duration of self-heating (days)</th>
<th>Air flow (L/min)</th>
<th>Maximum temperature (°C)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Moura D</td>
<td>77</td>
<td>93 - 450</td>
<td>56.0</td>
</tr>
<tr>
<td>Dartbrook 1 (NSW-D1)</td>
<td>36</td>
<td>56 - 403</td>
<td>36.2</td>
</tr>
<tr>
<td>Dartbrook 2 (NSW-D2)</td>
<td>51</td>
<td>155 - 334</td>
<td>63.9</td>
</tr>
</tbody>
</table>
Tests with a thermal cover

An insulated cover was installed over the reactor to reduce the heat losses to the environment and facilitate self-heating. Two samples of Dartbrook coal (NSW-D3 and NSW-D4), a Newcastle region coal (NSW-M1) and two Queensland coals from the Bowen Basin (QLD-BB1, QLD-BB2) were allowed to self-heat for varying periods with air flows as outlined in Table 2. Full details of these run of mine coals and the test results were previously reported as follows, Dartbrook coal samples NSW-D3 and NSW-D4 in the final report for ACARP Project C5031 (Cliff et al, 1998, 1999, 2000a, 2000b; Clarkson and Davis, 2000), NSW-M1 in Appendix A of ACARP Project C10015 (Clarkson, 2004), QLD-BB1 has not previously been reported. QLD-BB2 is to be included in the report for ACARP Project C12018.

The crushed Dartbrook 3 (NSW-D3) sample was allowed to self-heat without assistance from the buried heater element. Although the NSW-D3 sample exceeded both the maximum temperature of the NSW-D2 test and the so called ‘critical’ 70°C mark, achieving a temperature of 147.3°C after 131 days, the reaction did not proceed to thermal runway as had been predicted. Beyond this point the maximum coal pile temperature showed a stepped behaviour consistent with the heating migrating past the thermocouples towards the inlet coal face where the oxygen supply was less restricted than in bulk of the reactor (Figure 4). A peak temperature of 221.6°C was attained after 172 days when the hot spot migrated to within 1 m of the inlet face. The test was terminated at this point as the oxygen demand exceeded the volume able to be supplied by the blower units (Cliff et al, 1999).

The run of mine Dartbrook 4 (NSW-D4) sample was also allowed to self-heat and was terminated after 149 days due to the exponential increase in maximum pile temperature which reached the order of 25°C/h (Clarkson and Davis, 2000). In this instance, the 261.8°C hot spot generated approximately 1 m from the inlet and tended to migrate vertically rather than horizontally probably due to the heat flux warming the coal above the initial hot spot, in the presence of surplus air (Cliff et al, 2000b).

The removal of moisture from the coal was considered to be the rate-limiting step in the self-heating process for both these samples.

![Dartbrook Day](image)
Gas samples collected from up to ten ports including the exhaust in the NSW-D3 reactor test and 16 ports in the NSW-D4 test allowed the progress of both heatings to be monitored by GC. The effectiveness of the current mine gas indicators of spontaneous combustion were evaluated and compared to the small-scale tests (Figure 5). The results of these two tests are summarised in Table 3 (Cliff et al., 2000a).

The run of mine Bowen Basin sample, QLD-BB1, was allowed to self heat for 289 days attaining a maximum temperature of 100.7°C before being sealed in an attempt to slow the reaction for operational reasons. The reactor was re-ventilated 13 days later with an airflow of approximately 100 L/min. The peak coal pile temperature fell from 94.1°C to 71.6°C over the following 15 days (day 317). The test was terminated on day 352 at a maximum pile temperature of 76.7°C, as the coal appeared to be reaching an equilibrium temperature with the cover temperature. At the request of ACARP the buried heater element was used to stimulate the heating from day 371. Within two days of the buried heater element being set at 400°C, the temperature of the surrounding coal had increased from 55.4°C to 100°C. Approximately 400 mm upwind of the buried heater element the coal temperature only increased by 0.5°C over the same time period. Six days later (day 379) the thermocouple at this point registered a temperature of 94.6°C, which escalated over the next 11 hours to 251.8°C.

The run of mine Newcastle region sample, NSW-M1 was allowed to self-heat for an extended period of time. The test was terminated after 506 days when an exponential increase in maximum pile temperature to 435.2°C was observed as the heating burnt through the insulation on a thermocouple strand. The peak temperature rose from 256.9°C to 435.2°C in an hour when the intake airflow was increased to approximately 1200 L/min for a period of less than four hours. The hot spot in this case generated approximately 1.22 m from the inlet, being located on the northernmost thermocouple strand between the fourth and third thermocouples of the third layer. Coal samples retrieved from this area show signs of charring and loss of volatiles (Clarkson, 2004) (see Figure 6).

Compared to the previous tests, which had ‘point source’ heatings, the NSW-M1 heating involved a considerable volume of coal. Clarkson (2004) reported that in the NSW-D4 heating, the area involved at an advanced stage in the heating was

### TABLE 2
Details of self-heating, with a thermal cover.

<table>
<thead>
<tr>
<th>Coal</th>
<th>Duration of self heating (days)</th>
<th>Air flow (L/min)</th>
<th>Maximum temperature (°C)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dartbrook 3 (NSW-D3)</td>
<td>172</td>
<td>42 - 115</td>
<td>221.6</td>
</tr>
<tr>
<td>Dartbrook 4 (NSW-D4)</td>
<td>149</td>
<td>48 - 483</td>
<td>261.8</td>
</tr>
<tr>
<td>Bowen Basin 1 (QLD-BB1)</td>
<td>289</td>
<td>54 - 115</td>
<td>100.7</td>
</tr>
<tr>
<td>Newcastle Region (NSW-M1)</td>
<td>506</td>
<td>38 - 115, 150 - 1204</td>
<td>435.2</td>
</tr>
<tr>
<td>Bowen Basin 2 (QLD-BB2)</td>
<td>384</td>
<td>84 - 111</td>
<td>49.1</td>
</tr>
</tbody>
</table>

### TABLE 3
Summary of experimental results.

<table>
<thead>
<tr>
<th>Observation</th>
<th>Crushed sample</th>
<th>Run of mine sample</th>
</tr>
</thead>
<tbody>
<tr>
<td>Exponential temperature rise</td>
<td>No stepped increase initially</td>
<td>Yes</td>
</tr>
<tr>
<td>Time taken to initiate self-heating</td>
<td>172 days to thermal runaway</td>
<td>149 days to thermal runaway</td>
</tr>
<tr>
<td>Comparison with small-scale</td>
<td>Good at hot spot, deviation further away</td>
<td>Excellent correlation</td>
</tr>
<tr>
<td>Odour</td>
<td>Consistent with small-scale tests – not fire stink</td>
<td>Consistent with small-scale tests – not fire stink</td>
</tr>
<tr>
<td>CO make</td>
<td>Est 0.5 L/min at 120°C Est 1.0 L/min at 200°C</td>
<td>1.2 L/min at 120°C, 2.58 L/min at 200°C</td>
</tr>
<tr>
<td>Size of hot spot</td>
<td>Approx 400 mm in radius</td>
<td>Approx 400 mm in radius</td>
</tr>
<tr>
<td>Location of hot spot</td>
<td>Middle layer, at inlet face</td>
<td>Middle layer, 1 m from inlet</td>
</tr>
</tbody>
</table>

**Small Scale Testing**

FIG 5 - Comparison of Graham’s ratios from the Dartbrook 3 large-scale reactor test with small-scale (laboratory) testing results.
localised (ie the diameter of the area above 100°C was estimated to be 800 mm to 1200 mm and that above 200°C only a few hundred millimetres). In the case of the NSW-M1 coal, the area above 100°C was ellipsoid in shape with a width up to 1600 mm and length of up to 2400 mm. The area above 200°C was a ‘U’ shaped band and extended over at least 4 × 400 mm grids.

The coal in the reactor was inerted using nitrogen but although the peak temperature dropped to less than 260°C within a few hours of the nitrogen injection, 14 days later a large volume of the coal continued to exhibit temperatures ranging from 100°C to 179°C. Further injections of nitrogen into the reactor via the gas sampling ports had limited success in inerting the reactor. On day 530, water was eventually used to cool the coal, which still exhibited temperatures as high as 172°C even though the thermal cover had been removed as part of the initial inertisation and shut down procedure 25 days earlier.

Based on the results of the previous testing and a moderate to high rating from the small-scale tests, this sample had been expected to take eight to nine months to react. There are several factors, which could have influenced the heating time line. The seam from which this coal originated is now known to have a tight cleat structure and high compressive strength compared to the Queensland QLD-BB1 sample, which had a low reactivity from the small-scale tests. It is possible that the high compressive strength of the NSW-M1 coal reduced the rate at which oxygen was able to penetrate the coal thus reducing the rate of oxidation, which is the heat-generating step in a spontaneous combustion event.

However the major factor in the longer heating time is now believed to be the start temperature of the test. It has been reported by Clarkson (2004) that the high reactivity NSW coal samples took between three and nine days to reach a maximum temperature of 40°C while the low reactivity Queensland coal, QLD-BB1, took 15 days to reach 40°C. By comparison the NSW-M1 coal sample took 242 days, eight months to reach 40°C and a further 263 days to complete the test (Figure 4). The time from 40°C to completion was just under nine months, which correlates with the original eight to nine month timeframe expected for the test.

During the self-heating phase of the NSW-M1 sample monitoring of the gases within the reactor showed that no methane or ethane were present as seam gas after 71 days. Carbon dioxide was however present, increasing with distance from the inlet, measuring 0.11 per cent 1 m from the inlet and 0.34 per cent. 3.2 metres from the inlet. Carbon monoxide levels throughout the reactor were only a few ppm until the coal began to dry out. The carbon monoxide levels measured 1 m from the inlet began to rise from a background of 2 - 3 ppm after 100 - 125 days. Deeper into the pile at the gas ports located 2 m from the inlet, the levels of carbon monoxide were not observed to begin to rise for approximately 200 days.

The CO/CO2 and Graham’s ratios were again found to be the best indicators of the state of the heating. Loss of carbon monoxide due to conversion to carbon dioxide or reabsorption onto the coal as it traversed the coal pile results in the severity of the heating being underestimated when measured in the exhaust (Figure 7).

Comparison of the gas ratios from the large-scale test results with that of the small-scale tests shows reasonable agreement of the Grahams ratio from both tests although the large-scale test results tend to slightly overestimate the degree of heating (Clarkson, 2004). In the case of the large-scale CO/CO2 results, there is good agreement to approximately 100°C but thereafter the ratio severely underestimates the state of the heating (see Figure 8).
The run of mine Bowen Basin sample, QLD-BB2, was allowed to self-heat for 384 days attaining a maximum coal pile temperature of 49.1°C. Attempts were then made to further encourage the development of the heating by setting the cover temperature above that of the coal edge temperature. The temperature of the inline heater for the recirculated air system was raised by 16°C over the next four days resulting in an increase in the cover temperature from 48.5°C to 54.6°C over the 12 days to day 396. Over the period of day 384 to day 410, the maximum coal pile temperature rose to 53.9°C in response to the heating stimulus. The test was terminated at day 429 with a maximum pile temperature of 53.9°C.

The QLD-BB2 sample had been expected to take of the order of nine to ten months to self-heat as it had come from the same seam as the QLD-BB1 sample. Small-scale testing of the QLD-BB2 sample including the Proximate and Ultimate analysis had characterised the large-scale sample as having a coal profile and spontaneous combustion potential typical of the originating seam, hence the predicted ten month time frame. The initial start temperature of this test was 29.6°C, which was almost identical to that of the NSW-M1 sample. In this instance the sample took 60 days to reach the 40°C mark. The low start temperature appears to be critical in the prolonged heating time of both the NSW-M1 and QLD-BB2 samples. By comparison the NSW-D3, NSW-D4 and QLD-BB1 had start temperatures of 36.7°C, 31°C and 33.1°C respectively having been started in late summer whereas the NSW-M1 and QLD-BB2 samples were started in early summer and late spring respectively. The implications from this for an underground mine are that the ground temperature may significantly influence the tendency to spontaneously combust with increasing ground temperature increasing the likelihood that a given coal will self-heat.

CONCLUSIONS

Extrapolation of large-scale tests to the underground mine situation indicates that the ground temperature of the mine will significantly influence the tendency of the coal to spontaneously combust. Heatings may be localised involving only a few tonnes of coal however more extensive heatings can develop. These heatings will be harder to inert and keep inerted as the mass of hot coal acts as a long-term heat reservoir even after oxygen has been excluded due to the inherent insulating properties of the coal.

Graham’s ratio, CO/CO₂ ratio and CO make are the best indicators of the state of a heating. These ratios can however underestimate the state of the heating where monitoring is remote from the site of the heating.

Investigation of a potential heating should begin as soon as ratio levels significantly exceed the background levels. Using standard values as action levels is meaningless.

Significant changes in oxidation products such as carbon monoxide, hydrogen, ethane and ethylene may only be measurable at an advanced stage of the heating.

Goaf monitoring needs to be carried out as comprehensively as time and resources allow so that all potential heating sites are monitored as close to them as possible. Heatings can only occur where there is sufficient oxygen, the coal is dry and the heat balance is in favour of heat retention (Cliff et al., 2000b).

ACKNOWLEDGEMENTS

The contributions of Simtars staff past and present to this project is gratefully acknowledged. The permission of the Director of Simtars, Stewart Bell to publish this paper is also gratefully acknowledged. This work was supported by ACARP, Anglo Coal and the Centennial Coal Company Limited.

REFERENCES


The Use of Electronic Nose Devices for Coal Self-Heating Detection

F Clarkson

ABSTRACT

Electronic noses are being used in the processing industries and in fields such as environmental and occupational hygiene to detect a wide variety of problems from contaminated foods to toxic chemicals in soils. Recently, research has been conducted into the use of these devices for detecting spontaneous combustion in mines with the aim of producing a method for early detection of a developing heating. Gas samples were collected from both Simtars’ 16 m^3 large-scale spontaneous combustion reactor and the University of Queensland’s two-metre column. The samples were analysed using an electronic nose. GCMS and HPLC were used to identify some of the components present.

A number of issues both from a practical mine situation and a scientific standpoint have been identified that need to be addressed before these devices can be used to detect a developing heating. Aluminised bags routinely used in the mining industry to collect gas samples for analysis of the permanent gases are not suitable for use with electronic nose devices as they have a ‘fingerprint’ from the polymer lining that interferes with the coal heating fingerprint. Tedlar bags used for environmental gas sampling are also unsuitable due to their fingerprint. Gas samples can be collected in glass gas bulbs with aluminium seals and successfully analysed. However the aluminium foil used to seal the bulbs is a restricted item in underground mines. It was also found that the volatile organic fingerprint of the coal heating was extremely weak at temperatures below 130°C. Before a useful application can be developed, further investigation into the chemical species present in the off-gases from Australian coals is required.

INTRODUCTION

A natural consequence of the weathering of coal is that the oxidation process generates heat. If the heat cannot be dissipated, the coal temperature increases leading to an increased rate of oxidation. In instances where there is sufficient heat accumulation, the coal spontaneously combusts and will eventually burst into open fire if there is sufficient oxygen present.

The propensity of coal to spontaneously combust has long been a problem in the mining industry. Available records indicate that in New South Wales from 1960 to 1991 there were one hundred and twenty five incidents of spontaneous combustion reported to the Inspectorate while in Queensland there were 68 reported incidents from 1972 to 1994 (Cliff, Rowlands and Sleeman, 1996). The results of spontaneous combustion episodes, even in recent times have had both tragic and serious economic consequences for the mines involved.

The zNose 7100 Fast GC Analyser

A run of mine coal sample, NSW-M1, from a Newcastle region mine in New South Wales was obtained for the large-scale 16 m^3 spontaneous combustion test. Details of the 16 m^3 spontaneous combustion test are included in Appendix A of ACARP Project C10015 Detection of Heating of Coal at Low Temperatures. The project set out to develop available electronic nose technology and apply it to problems of early spontaneous combustion detection, so that a method could be established whereby mines could identify the onset of a spontaneous combustion, monitor its progress and initiate control strategies.

EXPERIMENTAL

A run of mine coal sample, NSW-M1, from a Newcastle region mine in New South Wales was obtained for the large-scale 16 m^3 spontaneous combustion test. Details of the 16 m^3 spontaneous combustion test are included in Appendix A of ACARP Project Report C10015 (Clarkson, 2004).

The zNose 7100 Fast GC Analyser

An Electronic Sensor Technology zNose Fast GC Analyser with a DB5 column and a ‘Saw’ sensor (Figure 1) was specified for the investigation of the spontaneous combustion heating fingerprint.

1. Simtars, PO Box 467, Goodna Qld 4300.
   Email: fiona.clarkson@nrm.qld.gov.au
The operation of the zNose was described by Hester and Clarkson (2003) as follows:

The Microsensor instrument is a microsized automatic thermal desorption gas chromatograph. A gas sample entering the instrument passes through a bed of absorbent material (known as the trap) that collects the analytes of interest. Flash heating of the trap liberates the volatile components onto the separating column in a very narrow band, after which they are separated from one another during the normal traverse of the column. At the end of the column a sophisticated microsensor detects the emergence of separated components from the column. The sensor contains a rapidly oscillating crystal and records the effect that the emerging components have on the oscillating frequency of the crystal while they momentarily adhere to the surface of the crystal. The first differential of the frequency versus time plot provides a reasonably familiar chromatogram.

As with classical gas chromatographs, the zNose identifies the presence of a particular component based on a characteristic known as the retention time (RT). The retention time is defined as the time at which the detector sees the maximum peak from the emerging component. As each method used to analyse a gas sample represents a unique set of operating conditions, the retention time at which the detector, in this case the microsensor, sees the emerging component was characteristic of that component on that instrument for a particular set of operating conditions.

In the case of the zNose instrument, a series of retention time characteristics of the components of interest were stored in a ‘Peak file’ linked to the analysis method and used to flag the presence of these components in the gas sample being analysed. The area under the peak was a measure of the quantity of the component adhering to the crystal surface of the sensor and could therefore be used to quantitate the analyte present in the gas sample. The zNose assigned an arbitrary unit of ‘Counts’ to the area under the peak (equivalent to the sensor’s frequency change) unless a scale factor and a defined unit (ie ppm) had been specified.

**Screening tests**

Initial screening tests of the coal gas sampling equipment including aluminised gas sample bags, Tedlar bags and glass gas bulbs were performed using the standard analysis method supplied with the instrument. The method known as 18ps-2.mth was used to determine the suitability of the sampling equipment currently in use in the mining industry and at Simtars for investigation of the coal heating fingerprint (Figures 2 - 4). The coal gas samples used to screen the suitability of the coal gas sampling equipment were obtained from Simtars’ small-scale gas evolution apparatus, the 16 m³ large-scale spontaneous combustion reactor and the University of Queensland (UQ) two-metre column. The 18ps-2.mth method comprised a column ramp rate of 18°C/s and sampling pump duration of five seconds.

**Coal testing – 16 m³ large-scale spontaneous combustion reactor**

Gas samples drawn from the New South Wales coal sample, NSW-M1, as it was being reacted in the Simtars’ 16 m³ large-scale spontaneous combustion reactor were also analysed using an optimised method R05 s20p20.mth where R05 defines the column ramp rate as 5°C/s and p20 was a sample pump duration of 20 seconds. This method also involved the use of a glass
gas-sampling bulb with an aluminium foil septa (Figure 5). The commercially available rubber and foil/rubber composite septa were found to contribute an unacceptably high background to the coal gas samples.

**RESULTS AND DISCUSSION**

**Screening tests**

Four litre aluminised bags are currently used in the mining industry to collect gas samples for permanent gas analysis (H₂, O₂, N₂, CH₄, CO, CO₂, C₅H₁₀, C₂H₆) by gas chromatograph. The aluminised bags exhibited a complex background fingerprint. The polymer used to line the aluminised bags and to manufacture the snap in component on which the sampling tube was mounted appeared to be the origin of this fingerprint. The sampling tube itself appeared to have almost no fingerprint (Table 1). The P5.80 peak in the sampling tube spectrum was probably due to residual contamination of the column by the odour component of the polymer lining.

Gas samples drawn from the coal sample, NSW-M1, as it was being reacted in the UQ two-metre column were analysed using the method 18ps-2.mth. In addition to the peaks known to be associated with the polymer components of aluminised bag, peaks at retention times of 1.66, 2.10, 2.42 and 2.95 seconds were identified in the NSW-M1 spectrum (Table 1).

The fingerprint obtained using the glass gas bulbs was less complex than that obtained using the aluminised bags (see Table 2). Peaks previously observed between a retention time of 3.66 to 4.62 seconds were absent indicating these peaks were part of the polymer fingerprint from the aluminised bags and not a background due to laboratory environment. The polymer related peaks P5.80 and P5.98 were reduced by at least a factor of 20. The component appearing at the P2.95 peak in the aluminised bag series was no longer present however the presence of a strong peak at the slightly longer retention time of 3.14 seconds could have been the same component. The retention times of the P1.66, P2.10, P2.42 components were not affected by changing the sampling vessel, indicating a probable coal origin. Some contamination due to a carryover of one of the Tedlar bag fingerprint components was also noted at the P2.84 peak. In addition, four new peaks at retention times of 1.85, 2.60, 4.82 and 5.06 seconds were identified in the glass bulb sampling system.

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In the Tedlar bag system, the P1.66, P2.42, P4.82 and P5.06 peaks appeared at similar temperatures to those identified in the glass tube system. The P2.60 and P3.14, which occurred either side of the Tedlar bag peak were not detected consistently and in the instances where they could be detected, the temperature tended to be higher. Inspection of the chromatograms indicated that this was because at a column ramp rate of 18°C per second, the adjacent component peaks were not discretely separated from the Tedlar P2.84 peak when present at low levels and therefore could not be detected by the peak detection software (Figure 7). The peak from the Tedlar bag component was of the order of hundreds of counts compared to the adjacent peaks, which were in the order of tens of counts.

Further the P1.85 peak was not detected in any of the coal gas samples and the P2.10 peak was detected over a wider temperature range than in the glass tube system.

Electronic Sensor Technology, the makers of the zNose instrument have reported that the zNose was able to detect both dimethyl acetamide and phenol as the primary contaminants in

### TABLE 1

Components identified in the aluminised gas sampling bags, method 18ps-2.mth.

<table>
<thead>
<tr>
<th>Peak table (18°C ramp)</th>
<th>RT (sec)</th>
<th>Counts</th>
<th>Counts</th>
<th>Counts</th>
<th>Counts</th>
</tr>
</thead>
<tbody>
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<td>Unused aluminised bag + instrument air</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plastic moulding only</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sampling tube only</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NSW-M1 maximum temp 81°C</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>P1.66</td>
<td>1.66</td>
<td>233</td>
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<tr>
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</tr>
<tr>
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<td>9</td>
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<td>3.96</td>
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<td>20</td>
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<td></td>
</tr>
<tr>
<td>4.82</td>
<td>36</td>
<td>59</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>5.06</td>
<td></td>
<td></td>
<td>64</td>
<td></td>
<td></td>
</tr>
<tr>
<td>5.32</td>
<td>635</td>
<td>161</td>
<td>30</td>
<td>107</td>
<td></td>
</tr>
<tr>
<td>P5.80 Aluminised bag</td>
<td>5.76</td>
<td>423</td>
<td>peak</td>
<td>93</td>
<td>1338</td>
</tr>
<tr>
<td>P5.98 Aluminised bag</td>
<td>5.98</td>
<td>2096</td>
<td>8268</td>
<td></td>
<td></td>
</tr>
<tr>
<td>6.78</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>P6.12 Rotor peak</td>
<td>8.12</td>
<td>1134</td>
<td>319</td>
<td>91</td>
<td></td>
</tr>
</tbody>
</table>

RT = retention time.

### TABLE 2

Heating fingerprint for NSW-M1 (Newcastle region), method 18ps-2.mth.

<table>
<thead>
<tr>
<th>Peak table (18°C ramp)</th>
<th>RT (sec)</th>
<th>Glass gas sampling bulbs</th>
<th>Tedlar bags</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Appearance temperature (°C)</td>
<td>Appearance temperature (°C)</td>
</tr>
<tr>
<td>P1.66</td>
<td>1.66</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>P1.85</td>
<td>1.85</td>
<td>80</td>
<td>ND</td>
</tr>
<tr>
<td>P2.10</td>
<td>2.10</td>
<td>80</td>
<td>20</td>
</tr>
<tr>
<td>P2.42</td>
<td>2.42</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>P2.60</td>
<td>2.60</td>
<td>100</td>
<td>150</td>
</tr>
<tr>
<td>P3.14</td>
<td>3.14</td>
<td>60</td>
<td>80</td>
</tr>
<tr>
<td>4.82</td>
<td></td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>5.06</td>
<td></td>
<td>20</td>
<td>20</td>
</tr>
</tbody>
</table>

ND = not detected.

RT = retention time.

FIG 7 - NSW-M1 gas sample collected at 66°C.
Tedlar bags (Electronic Sensor Technology, nd). They further reported that even after flushing the bags ten times with clean nitrogen, the Tedlar bags continued to outgas these two contaminants at parts per million levels.

**Coal testing – 16 m³ large-scale spontaneous combustion reactor**

Following optimisation and calibration of the zNose 7100 instrument, a fingerprint peak file R05-s20p20.pkd comprising 21 components was constructed based on the fingerprint for a coal sample heated at 350°C (Table 3). The per cent spread (PCT spread) defines the retention time window within which the zNose will identify a peak as a particular substance.

Analysis of the port 4 and 5 gas samples using the method R05-s20p20.mth found that no components (ie volatile organic compounds (VOCs)) corresponding to the fingerprint defined in the R05-s20p20 peak file were reliably detected below 87°C. The P8-DW and the singlet-doublet peak component (4.04) were the first fingerprint components detected and had a temperature range of 87°C to 189°C and 87°C to plus 400°C respectively (Figures 8 - 10). Two other fingerprint components were detected at 92°C, namely the P6-DW peak and the P10-DW peak. The concentration of these components together with that of the P8-DW component appeared to increase with temperature (Figure 11) indicating these components were either products of the oxidation of coal or were volatile organic compounds tightly bound to the coal surface which desorbed as the coal self-heats.
At higher temperatures, a number of other VOC components were detected with retention times of less than 3.76 seconds (P6-DW component) (Figure 9). Given that the instrument separates and therefore detects VOC compounds on the basis of their volatility, ie the smaller the retention time the more volatile the compound, it can be inferred that these components were smaller more volatile organic species than those occurring at the P6-DW, singlet-doublet (4.04), P8-DW, and P10-DW retention times. It was also possible that these more volatile compounds were products of incomplete oxidation of the coal.

The VOC components with retention times of greater than 5.60 did not appear to be related to the low temperature heating of the NSW-M1 coal as the detected concentrations did not change or trend significantly with temperature. The appearance temperatures of the VOC components analysed using the method R05-s20p20.mth are summarised in Table 4.

<table>
<thead>
<tr>
<th>Substance</th>
<th>RT (sec)</th>
<th>PCT spread</th>
</tr>
</thead>
<tbody>
<tr>
<td>P1-DW, 2.16</td>
<td>2.16</td>
<td>2.5</td>
</tr>
<tr>
<td>P2-Toluene, 2.4</td>
<td>2.40</td>
<td>4.0</td>
</tr>
<tr>
<td>P3-DW, 2.74</td>
<td>2.74</td>
<td>2.5</td>
</tr>
<tr>
<td>P4-DW, 3.14</td>
<td>3.14</td>
<td>2.5</td>
</tr>
<tr>
<td>P5-DW, 3.46</td>
<td>3.46</td>
<td>3.5</td>
</tr>
<tr>
<td>P6-DW, 3.76</td>
<td>3.76</td>
<td>2.5</td>
</tr>
<tr>
<td>P7-DW, 4.34</td>
<td>4.34</td>
<td>2.5</td>
</tr>
<tr>
<td>P8-DW, 4.70</td>
<td>4.70</td>
<td>2.5</td>
</tr>
<tr>
<td>P9-DW, 5.04</td>
<td>5.04</td>
<td>2.5</td>
</tr>
<tr>
<td>P10-DW, 5.44</td>
<td>5.44</td>
<td>2.5</td>
</tr>
<tr>
<td>P11-DW, 5.84</td>
<td>5.84</td>
<td>2.3</td>
</tr>
<tr>
<td>P12-DW, 6.12</td>
<td>6.12</td>
<td>2.3</td>
</tr>
<tr>
<td>P13-DW, 6.80</td>
<td>6.80</td>
<td>1.5</td>
</tr>
<tr>
<td>P14-DW, 7.02</td>
<td>7.02</td>
<td>1.5</td>
</tr>
<tr>
<td>P15-DW, 7.9</td>
<td>7.90</td>
<td>2.5</td>
</tr>
<tr>
<td>P16-DW, 8.60</td>
<td>8.60</td>
<td>2.5</td>
</tr>
<tr>
<td>P17-DW, 9.10</td>
<td>9.10</td>
<td>2.5</td>
</tr>
<tr>
<td>P18-DW, 9.72</td>
<td>9.72</td>
<td>2.5</td>
</tr>
<tr>
<td>P19-DW, 10.41</td>
<td>10.41</td>
<td>2.5</td>
</tr>
<tr>
<td>P20-DW, 11.06</td>
<td>11.06</td>
<td>2.5</td>
</tr>
<tr>
<td>P21-DW, 11.64</td>
<td>11.64</td>
<td>2.2</td>
</tr>
</tbody>
</table>

PCT spread = per cent spread
RT = retention time.

At higher temperatures, a number of other VOC components were detected with retention times of less than 3.76 seconds (P6-DW component) (Figure 9). Given that the instrument separates and therefore detects VOC compounds on the basis of their volatility, ie the smaller the retention time the more volatile the compound, it can be inferred that these components were smaller more volatile organic species than those occurring at the P6-DW, singlet-doublet (4.04), P8-DW, and P10-DW retention times. It was also possible that these more volatile compounds were products of incomplete oxidation of the coal.

The VOC components with retention times of greater than 5.60 did not appear to be related to the low temperature heating of the NSW-M1 coal as the detected concentrations did not change or trend significantly with temperature. The appearance temperatures of the VOC components analysed using the method R05-s20p20.mth are summarised in Table 4.
Gas chromatography – mass spectrometry and high performance liquid chromatography analysis

The gas chromatography-mass spectrometry (GC-MS) analysis of the off gases from the 16 m³ spontaneous combustion reactor indicated the presence of C6 - C9 alkanes, several ketones and aromatic benzene compounds below 100°C (Table 5). The alkane and ketone compounds appeared to increase in concentration with increasing temperature below 100°C although there was some uncertainty as to the size of the increase for the hexane and heptane samples due to the high sample concentration on the ATD tubes. With the exception of benzene, the aromatic benzene type compounds did not appear to increase with temperature below 100°C. The higher concentrations of the alkane, ketones and aromatic benzene compounds measured in the exhaust port gases compared to ports 4 and 5 indicate that these compounds were being produced over a wider area than just the port 4 and 5 locality and that their participation in absorption or secondary reactions within the coal pile was limited. Therefore, these compounds were stable enough to be expected to be released into the goaf air. The stability of these compounds once in the goaf type environment was not determined.

The high performance liquid chromatography (HPLC) analysis of the off gases from the 16 m³ spontaneous combustion reactor indicated the presence of a number of carbonyl components (oxygenated species) in the off-gases (Table 6). The presence of formaldehyde appeared to be temperature dependent, increasing in concentration between 75.8°C and 123.5°C. The temperature dependence of the evolution of acetaldehyde, acetone, propionaldehyde and the heavier more complex oxygenated species was not clear from the current data.

The lower concentrations of acetaldehyde, acetone and propionaldehyde in the exhaust port gases indicated these oxygenated species were probably being absorbed within the coal pile to some degree. The potential for these gases to be found in the goaf was therefore limited. However if they were detected in goaf gas samples, this could indicate that the site of the heating was close to the sampling point. The higher levels of formaldehyde in the exhaust port were most likely due to the fact that it is a small highly volatile compound and, therefore, did not readily absorb onto the coal as it traversed the coal pile.

Overall the aromatic hydrocarbons and, to a limited degree the oxygenated forms of the volatile organic compounds, appeared to coexist rather than one in preference to the other.

### Table 5

<table>
<thead>
<tr>
<th>Reactor max temp (°C)</th>
<th>75.8</th>
<th>86.3</th>
<th>86.3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Port</td>
<td>4 and 5</td>
<td>4 and 5</td>
<td>Exhaust</td>
</tr>
<tr>
<td>Compound name</td>
<td>Amount (µg/L)</td>
<td>Amount (µg/L)</td>
<td>Amount (µg/L)</td>
</tr>
<tr>
<td>Dichloromethane</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>Methyl ethyl ketone</td>
<td>0.079</td>
<td>0.282</td>
<td>0.449</td>
</tr>
<tr>
<td>Ethyl Acetate</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>n-Hexane</td>
<td>0.197&lt;sup&gt;†&lt;/sup&gt;</td>
<td>0.807&lt;sup&gt;†&lt;/sup&gt;</td>
<td>1.061&lt;sup&gt;†&lt;/sup&gt;</td>
</tr>
<tr>
<td>Chloroform</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>1,1,1-Trichloroethane</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>Benzene</td>
<td>0.014</td>
<td>0.039</td>
<td>0.057</td>
</tr>
<tr>
<td>Trichloroethene</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>n-Heptane</td>
<td>0.500&lt;sup&gt;†&lt;/sup&gt;</td>
<td>0.780&lt;sup&gt;†&lt;/sup&gt;</td>
<td>1.489&lt;sup&gt;†&lt;/sup&gt;</td>
</tr>
<tr>
<td>Methyl isobutyl ketone</td>
<td>0.010</td>
<td>0.015</td>
<td>0.024</td>
</tr>
<tr>
<td>Methyl cyclohexane</td>
<td>0.309</td>
<td>0.578</td>
<td>0.796</td>
</tr>
<tr>
<td>1,1,2-Trichloroethane</td>
<td>0.001</td>
<td>0.003</td>
<td>0.003</td>
</tr>
<tr>
<td>Toluene</td>
<td>0.016</td>
<td>&lt;RL</td>
<td>0.020</td>
</tr>
<tr>
<td>n-Octane</td>
<td>0.184</td>
<td>0.309</td>
<td>0.551</td>
</tr>
<tr>
<td>Tetrachloroethylene</td>
<td>0.001</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>1,1,1,2-Tetrachloroethane</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>Ethyl benzene</td>
<td>0.012</td>
<td>0.002</td>
<td>0.017</td>
</tr>
<tr>
<td>m- and p-Xylene</td>
<td>0.004</td>
<td>0.005</td>
<td>0.010</td>
</tr>
<tr>
<td>1,1,2,2-Tetrachloroethane</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>2-Butoxyethanol</td>
<td>0.001</td>
<td>&lt;RL</td>
<td>0.003</td>
</tr>
<tr>
<td>o-Xylene</td>
<td>0.003</td>
<td>0.003</td>
<td>0.007</td>
</tr>
<tr>
<td>n-Nonane</td>
<td>0.064</td>
<td>0.112</td>
<td>0.188</td>
</tr>
<tr>
<td>1,3,5-Trimethylbenzene</td>
<td>0.002</td>
<td>0.002</td>
<td>0.004</td>
</tr>
<tr>
<td>1,2,4-Trimethylbenzene</td>
<td>0.002</td>
<td>0.003</td>
<td>0.005</td>
</tr>
<tr>
<td>n-Decane</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
<td>&lt;RL</td>
</tr>
<tr>
<td>1,2,3-Trimethylbenzene</td>
<td>0.001</td>
<td>0.001</td>
<td>0.002</td>
</tr>
</tbody>
</table>

<sup>†</sup> Results indicated with † were above the calibration range.

**Fig 11** - Dependence of NSW-M1 VOC components on coal temperature.
Implications for applying electronic nose technology to detection of heating of coal at low temperatures

While it was possible to define a fingerprint that could be used to characterise the heating profile of a coal sample, the main discrimination achievable was that an advanced heating did or did not exist.

It is clear from the zNose results that the levels of VOCs produced by a low temperature heating (ie less than 100°C) are very small compared to other sources of VOCs present in gas sampling media. This includes media such as the aluminiumised bags and Tedlar bags which are routinely used to handle coal gas samples. In order to obtain a reasonable baseline to enable detection of the VOC components of the coal gas spectrum, it is necessary to go to inordinate lengths to identify and remove items from the sample handling and collecting system which have a characteristic VOC profile of their own. This is important as the higher levels of background VOCs either overlap the coal VOC spectrum or overload the instrument column and/or sensor before a useful loading of coal VOC components can be achieved. Maintaining a ‘clean’ sampling system in the mine situation is much more difficult than in a controlled laboratory environment.

Further as a result of the contamination issues, gas samples could only be collected in glass gas bulbs and vials which had to be sealed with an aluminium foil septa. The commercially available rubber and foil/rubber composite septa were found to contribute an unacceptably high background to the coal gas samples. A suitable substitute for the aluminium foil septa was not identified during the project. This would need to be done if further work was undertaken as the use of aluminium is restricted in underground coal mines.

The low levels of VOCs present in the coal signature below 100°C would also represent a significant challenge in detecting a low temperature heating in the mining environment as, in all likelihood, VOC emissions from diesels, plastics and electrical equipment used in the mine could produce a significant background of their own and swamp the coal VOC signature. Even the electronic noses using metal oxide, conducting polymer and quartz microbalance sensors which respond to a wider range of compounds could be compromised by the ‘uncontrolled’ mine environment. Potentially they could return either a false positive or exhibit so little change on a contaminant loaded sensor that no measurable change might register until the heating was at an advanced stage. It is not possible to assess the potential sources of contamination or their ability to bias sensor readings without knowing more about the nature of the low temperature coal emissions, the mining environment and the sensor’s affinity for specific compounds.

The major implication from all these factors is that a better understanding of the range of volatile compounds that are or are not present from low temperature coal heating is required together with a knowledge of the compounds routinely found in the mine atmosphere. This is necessary in order to determine if electronic noses or even other analytical techniques can be applied to the detection of a low temperature heating.

In particular there is a need to determine the presence or otherwise of:
1. oxygenated species such as acetone, methanol, formaldehyde,
2. volatile organic compounds such as benzene, toluene, hydrocarbons, and
3. sulfur compounds.

This knowledge will then allow an assessment to be made as to whether, besides the currently used CO and CO₂ profiles, there exist any gases below 100°C that have the potential to be used as an indicator of a low temperature heating. The limited GC-MS and HPLC work undertaken as part of this project has confirmed that some oxygenated and VOCs do exist in coal heated at less than 100°C however there is insufficient data available to determine if they have a potential to be used as heating indicators.

CONCLUSIONS

Very low levels of volatile organic compounds are emitted from coal below 100°C. Alkanes C₆ - C₉, several ketones, aromatic benzene compounds, acetone, formaldehyde and several other oxygenated compounds are present in gas samples from coal reacting below 100°C. The low temperature range over which these volatiles are emitted from coal is not well described.

The zNose instrument has shown some capacity to detect a heating below 100°C but this ability was easily compromised by contaminants in the sampling system. Aluminised gas sample bags and Tedlar bags give off significant levels of volatile organic compounds, which contaminated the coal gas samples. Use of glass gas sample bulbs and vials sealed with aluminium foil was necessary to avoid contamination of the coal gas samples. The zNose was able to discriminate as to whether a heating is at an advanced stage or not. Hence, the zNose instrument was capable of providing detailed fingerprints of gases evolved during coal heating once an advanced (ie pyrolysis) stage is reached.

A better understanding of the range of volatile compounds that are or are not present from low temperature coal heating is required. This together with a knowledge of the compounds routinely found in the mine atmosphere will determine if electronic noses or even other analytical techniques can be applied to the detection of a low temperature heating.

ACKNOWLEDGEMENTS

The contributions of Simtars staff past and present to this project is gratefully acknowledged. The permission of the director of Simtars, Stewart Bell to publish this paper is also gratefully acknowledged. This work was supported by ACARP and Centennial Coal Company Limited.
REFERENCES


The Ability of Current Gas Monitoring Techniques to Adequately Detect Spontaneous Combustion

D Cliff

ABSTRACT

This paper investigates the adequacy of current gas monitoring techniques to adequately detect spontaneous combustion in underground coalmines. Despite being in the 21st century spontaneous combustion continues to occur in underground coalmines sometimes being detected only at a very advanced stage. Control of the incident is often then very expensive and time consuming.

The adequacy needs to be assessed not only from the point of view of the analysis technique be it tube bundle, gas chromatograph or real time sensor but also the number, location and sampling frequency of the monitoring locations.

Recommendations are made to optimise monitoring processes and recognise limitations of current techniques.

INTRODUCTION

There are three key questions that need to be addressed when designing a mine environment monitoring system:

• What are you trying to monitor?
• Where are you going to monitor? and
• How are you going to monitor?

Determining the answer to the first question will define the boundary conditions for defining the answers to the second and third questions.

WHAT TO MONITOR?

The focus of this paper is monitoring for the detection of spontaneous combustion, however mines are required to monitor for a range of situations including, safe working conditions for the workers, outburst prevention, equipment fires and statutory monitoring requirements, eg ERZ in Queensland.

Classically monitoring for the detection of spontaneous combustion has focussed on gas monitoring and exceeding of predetermined maximum allowable values for gas concentrations (eg carbon monoxide) or derived indicators such as Graham’s ratio. Most of these indicators have been derived based upon either laboratory testing or events in underground coalmines that occurred many years ago. Often the conditions in these mines bore no similarity to modern underground coalmines. Historically for example, spontaneous combustion events would often occur in the pillars of roadways and were detected by smell or a rise in CO make. Now the majority of incidents occur in the goaf of a longwall panel some distance behind the face. The heating was detected when gas samples were taken through a seal into the goaf that revealed abnormal CO and H2 concentrations. Initially there was little indication of the location of the actual heating.

In each of these cases a heating developed in the goaf of a longwall panel some distance behind the face. The heating was detected when gas samples were taken through a seal into the goaf that revealed abnormal CO and H2 concentrations. Initially there was little indication of the location of the actual heating.

In two of the cases the application of inert gas into the goaf controlled the heating. Unfortunately in the third case the heating developed so rapidly that it became a raging fire and sealing at the surface was the only option, after inertisation was tried.

The gestation period of the heating in each case is unknown except that a maximum value can be established from the time the goaf was established. In two cases there was no indication of a worsening situation, in part due to the absence of regular gas monitoring through the seals. Local conditions, such as water blocking access to the seal prevented sampling in one case.

In one case it was only after sampling from a line of seals that it was determined that the heating was remote from the original goaf. The treatment of the heating was protracted and it is likely that several lesser intensity oxidation occurrences initiated subsequent to the original heating.

For another case after sampling along the gate road into the goaf at various seal locations, the seat of the heating was determined to be close to a particular gate-road seal, and a surface borehole was able to intersect the heating allowing the application of inert gas directly onto it.

In the third case there is still today no definite evidence to locate the source of the heating. In each case however there is no way of knowing the genesis of the heatings in terms of what caused that particular area of goaf coal to abnormally oxidise and not the millions of tonnes of other coal in the goaf all around it. Circumstances at that point must just have been right for it to propagate. The initiation of the event in each case probably occurred months beforehand and the oxidation stewed away until conditionsfavouredacceleration. In two of the cases this was caused by sudden influx of additional air due to seal failures. In the third case it was probably simply a case of the longwall had been stationary for a number of weeks and air was able to continually flow to the heating site, under conditions that favoured abnormal oxidation.

Spontaneous combustion is a complex process and the chemistry of the process is still not well characterised. Laboratory experiments at SIMTARS (see for example Cliff et al., 2000) and UQ (Beamish, Barakat and St George, 2001) clearly show the complexity involved when coal reacts with air. Figure 1 depicts a ‘typical’ bituminous coal molecule. Coal is of course not a simple molecule rather it is a complex mixture of a range of large organic molecules containing carbon, oxygen, hydrogen, nitrogen and sulfur. Add to this impurities such as carbonates, pyrites and salts, stir in seam gases (methane and/or carbon dioxide) and water and you get coal as we know it. Some parts of the coal are far more reactive than others.

For example when methane is oxidised to carbon dioxide it goes through a series of intermediate compounds – methanol to formaldehyde to formic acid to carbon dioxide.

\[
\text{CH}_4 - \text{CH}_3\text{OH} - \text{HCHO} - \text{HCOOH} - \text{CO}_2
\]

The hardest step to achieve is the first step; methane is very unreactive and needs a lot of help (energy and catalysis) to begin.

REFERENCES

1. MAusIMM, Minerals Industry Safety and Health Centre, Sustainable Minerals Institute, The University of Queensland, Brisbane Qld 4072.
the process. The further to the right the process proceeds the easier it becomes. Thus acid functional groups are very reactive and alkyl functional groups are not. In Figure 1, the bits around the molecule labelled – CHx are thus unreactive whereas those containing oxygen are more reactive. Not surprisingly low rank coals contain more of the oxygenated species than high rank coals and hence have a higher inherent reactivity.

Of course being a mixture of many chemical components it means that the oxidation chemistry is also complex. Figure 2 below indicates a simplified model of the oxidation process. Even in this model each reaction step has its own temperature dependence as well as individual dependence on the concentration of the reactants. Coal reacts as a solid and thus the effective surface area available for reaction is an important factor. If oxygen cannot get into the coal to reach the reactive components of the coal then reaction cannot occur. In other words the presence of water and seam gas within the pores of the coal reduces the effective surface area of the coal available to react and hence the potential for the coal to heat up and proceed to spontaneous combustion is also reduced.

Similarly if the most reactive components of the coal macromolecule have already reacted, then the rate of oxidation is substantially reduced, ie if a coal has been exposed to air for a long time, the reactive components will have reacted and the heat will have dissipated to the atmosphere, the residual ‘weathered’ coal will be unreactive. A more detailed description of the chemistry of coal oxidation can be found in Cliff and Bofinger (1998).

Figure 3 illustrates two tests carried out on Dartbrook coal samples in the large-scale (16 tonne) reactor at SIMTARS (Cliff et al, 2000). It can clearly be seen that the coal apparently lies dormant for many days and then suddenly the oxidation process accelerates out of control in a few hours. This translates to negligible gas concentrations and ratios suddenly becoming huge. In the case of the run of mine test, the CO make went from less than 1 L/min to over 100 L/min in less than 24 hours.

This is consistent with the laboratory observation that for every ten degrees increase in reaction temperature there is a doubling in the nett reaction rate and thus gas evolution rate. The dormant period appears to align with the dehydration of the coal and thus the energy being generated by the oxidation process is being absorbed by the energy requirements to volatilise the water out of the coal. Once this process is complete then the energy is channelled instead into heating the coal.

What this all means is that given the difficulty in detecting an active heating we should focus on preventing a heating from occurring. Monitoring strategies defined by an early response should be triggered by such things as:

- The detection of oxygen in areas of the goaf where it should not be. This does not immediately cause trouble but it will be the catalyst if this condition remains in place for any length of time. Remedial action to reduce the oxygen supply can avert a heating. Such action could include proactive inertisation, tightening of seals and reducing the pressure difference across the face of the longwall.
- The ability of oxygen to pass into areas of particular coal in the goaf for longer than normal, eg if the longwall stops for any length of time or is reduced to slow production rates.
- Pressure differences across seals that are not what is expected – this of course presumes that you know what to expect. Abnormal pressures differences often indicate leaking seals and air ingress into goaf areas.
The detection of unusual trends in gas behaviour particularly carbon monoxide. For example, generally as the longwall retreats away from a gate road seal, the oxidation behaviour of the coal goes through a maximum and tails away, due to the reduced availability of oxygen and the weathering of the coal. A peak in CO is typically seen one or two cut throughs behind the face. If a peak is found further back or the smooth trend behaviour from seal to seal is not observed then something unusual is happening.

Knowing the temperature profile across the face and around the goaf so that abnormal temperatures can be identified early. Temperature is very important as it affects the reaction rates as discussed above.

In the past too much reliance has been placed on small-scale laboratory testing such as R70 determinations, or in gas evolution testing. These have limited application to the real world, as the laboratory conditions bear no resemblance to those conditions found in a longwall goaf. R70 tests for example are carried out on small samples of dried crushed coal, which has been degassed. Often the tests are carried out under conditions where the airflow through the sample is much higher than would be found in the goaf. This means that the balance of reaction mechanisms depicted in Figure 2 above will differ from that in the goaf and hence give different gas evolution behaviour.

Medium and large-scale testing using run of mine coal and more realistic airflows often give very different results. For example the two-metre column work of Beamish et al (2002 and 2003), has been able to demonstrate that significant levels of hydrogen can be generated at temperatures much less than 100°C. These results are consistent with an oxidation pathway where coal reacts with air in the presence of water vapour, and internally rearranges itself to generate carbon dioxide and hydrogen. A separate oxidation pathway appears to generate a mixture of carbon monoxide and carbon dioxide.

WHERE TO MONITOR?

Monitoring needs to be undertaken to ensure that normal behaviour can be characterised and that abnormality can be detected as soon as possible. Too often mines collect inadequate data.
amounts of data from far too few monitoring locations. This means that when something abnormal is detected, the situation is often serious and evacuation of the mine is the only option. Typical goaf seal behaviour needs to be determined including:

- pressure differentials across seals and around goafs as a function of distance from the face and other factors such as the change in the pressure difference across the face;
- gas evolution and derived indicators as a function of distance from the face, this is especially important where factors such as goaf drainage and or back bye ventilation is used to reduce seam gas impacts on the face; and
- longwall return concentrations and derived indicators such as CO make as a function of operating parameters including size of goaf, rate of retreat, etc.

Monitoring is a multi-tier process. Initially characterisation of normal goaf and return roadway behaviour will be an intensive campaign until sufficient data has been gathered to give the mine confidence that it knows what to expect. Once this initial characterisation is complete then the monitoring can be tailored to check that the expected behaviour is occurring.

HOW TO MONITOR?

Frequency and complexity of sampling will depend on what is being monitored. Continuous monitoring of panel returns is required by regulation. Seal sampling can be undertaken by a mix of tube bundle sampling with analysis by a bank of infrared analysers for CO, CO₂ and CH₄, paramagnetic for O₂, and bag samples with analysis by gas chromatograph for the seals immediately inbye the face and perhaps just bag samples taken on a regular basis for those seals toward the rear of the panel or if there is spare tube bundle capacity, these tubes could be sampled less frequently than the more important (more likely to change) tubes. It may not be necessary to sample every seal if there is no indication of abnormality in terms of pressure differentials or oxygen presence. Gas chromatography gives the opportunity to directly analyse for the presence of hydrogen and higher hydrocarbons as well as check the accuracy of the tube bundle system. The older the coal the harder in general it is to resuscitate it for abnormal oxidation.

DETECTION

Much has been written about the use of indicators for detecting spontaneous combustion. There is only one real law for spontaneous combustion monitoring – there is no universal indicator.

There are many indicators that have been used to detect spontaneous combustion including:

- Carbon monoxide concentration. This is very unreliable in isolation as CO is produced at all temperatures by coal oxidation and extensive oxidation over a long time may well produce the same concentration as a small much more intensive oxidation event.
- Hydrogen concentration. This is also unreliable in isolation as the work of Beamish et al (2003) and Nehemia, Davidi and Cohen (1999) have demonstrated.
- Ethane. Ethane is most commonly present as a minor seam gas typically between 1/100 and 1/1000 of the methane present. Do not use it as an indicator of spontaneous combustion.
- Ethylene concentration. At high temperatures (> 200°C) the coal will pyrolyse and produce a raft of unsaturated and saturated hydrocarbons. The presence of significant concentrations of ethylene (>10 ppm) is a reliable indication that abnormal oxidation has occurred. Typically by the time this occurs the CO and hydrogen concentrations are orders of magnitude higher than this and rising.
- CO make. This is only valid in roadways with defined, known ventilation. Absolute numbers only have meaning when they are calibrated against the actual mine performance and operating conditions (see above). Real time air velocity/air quantity sensors are readily available and allow for real time monitoring of the ventilation flows as well as determining makes.
- Graham’s ratio. This ratio can be a useful indicator of advanced oxidation however it is possible that effects of a small intense heating will be hidden in the effects of a large-scale low-level oxidation. This will cause Graham’s ratio to underestimate the intensity of any oxidation process.
- CO/CO₂ ratio. This ratio suffers from the same problems as Graham’s ratio and also interference from any CO₂ that is present in the seam gas.

There are many other indicators that have been suggested but none of them offer anything significant to the above, as they suffer from the same problems (Cliff, Hester and Bofinger, 1999) and may only serve to confuse the diagnosis. Some indicators such as CH₄ to CO₂ can be used to help identify anomalies and locate leaking seals or abnormal ventilation circuits in goafs.

Care needs to be taken when using inertsation techniques as they upset the parameters on which a number of computer programs calculate ratios as they use preset factors for such things as the ratio of oxygen to nitrogen in inlet air. This in turn will invalidate oxygen deficiency calculations and distort Graham’s ratio, artificially reducing it. Equally importantly introducing additional flows into goafs can cause existing flow paths to alter and direct different gas atmospheres to monitoring locations with consequent effects on the interpretation.

Trigger action response plans (TARPS) for spontaneous combustion should be established to initiate when something abnormal is detected, hopefully indicating a precursor to advanced oxidation rather than actual advanced oxidation. This allows preventative action to be initiated without impacting on the production of the mine. TARPS should not simply be for evacuation or major concern, they should initially be advisory and necessitate action by perhaps just the ventilation officer and his support crew. The response to a trigger should be appropriate to the risk the trigger reflects. Why evacuate the mine when CO exceeds 100 ppm in the goaf? Is there a flammable atmosphere there? What is the source of the CO – is it extensive oxidation or intensive? These are questions that would modify the response to the trigger. In this modern era there is no need to use simple triggers relying on the measurement of one gas. Mine environment monitoring systems are capable of providing a lot of information and it should be utilised to assist in the decision making process.

CONCLUSION

In summary, prevention is better than cure, especially where there is no guarantee that a heating can be detected at a stage early enough to control it quickly and easily.

Comprehensive monitoring systems need to be established to establish normal mine environment behaviour and understand the factors that can affect gas concentrations in all areas of the mine, including monitoring pressure differences around the mine, air flows and temperatures. Proper maintenance and personnel skilled in understanding mine monitoring systems and interpretation of mine atmospheres must support these systems.

REFERENCES


Spontaneous Combustion and Simulation of Mine Fires and Their Effects on Mine Ventilation Systems

A D S Gillies1, H W Wu2 and D Humphreys3

ABSTRACT
The structure of a comprehensive research project into mine fires study applying the Ventgraph mine fire simulation software, preplanning of escape scenarios and general interaction with rescue responses is outlined. The project has Australian Coal Association Research Program (ACARP) funding and also relies on substantial mining company site support. This practical input from mine operators is essential and allows the approach to be introduced in the most creditable way. The effort is built around the introduction of fire simulation computer software to the Australian mining industry and the consequent modelling of fire scenarios in selected different mine layouts.

Application of the simulation software package to the changing mine layouts requires experience to achieve realistic outcomes. Most Australian mines of size currently use a ventilation network simulation program. Under the project a small subroutine has been written to transfer the input data from the existing mine ventilation network simulation program to ‘Ventgraph’. This has been tested successfully. To understand fire simulation behaviour on the mine ventilation system, it is necessary to understand the possible effects of mine fires on various mine ventilation systems correctly first. Case studies demonstrating the possible effects of fires on some typical Australian coal mine ventilation circuits have been examined. The situation in which there is some gas make at the face and effects with fire have also been developed to emphasise how unstable and dangerous situations may arise.

The primary objective of the part of the study described in this paper is to use mine fire simulation software to gain better understanding of how spontaneous combustion initiated fires can interact with the complex ventilation behaviour underground during a substantial fire. It focuses on the simulation of spontaneous combustion sourced heatings that develop into open fires. Further, it examines ventilation behaviour effects of spontaneous combustion initiated pillar fires and examines the difficulties these can be present if a ventilation reversal occurs. It also briefly examines simulation of use of the inertisation to assist in mine recovery.

Mine fires are recognised across the world as a major hazard issue. New approaches allowing improvement in understanding their consequences have been developed as an aid in handling this complex area.

INTRODUCTION
Many people consider that mine fires remain among the most serious hazards in underground mining. The threat fire presents depends on aspects such as the nature and amount of flammable material, the ventilation system arrangement, the duration of the fire, the extent of the spread of combustion products, the ignition location and the reaction of personnel present.

An Australian Coal Association Research Program supported project incorporating a number of mine site exercises, as described by Gillies, Wala and Wu, 2004 and Wu, Gillies and Wala, 2004 has been undertaken focused on the application of mine fire and ventilation software packages for contaminant tracing and fire modelling in coal mines. This paper in particular examines aspects of spontaneous combustion initiated open fires in underground workings.

The study into this complex area has utilised the recently upgraded Polish mine fire simulation software, ‘Ventgraph’. There is a need to understand the theory behind the simulation program and to allow mine site use by those already familiar with the main existing mine ventilation analysis computer programs currently popular within the Australian, United States and South African industries such as ‘Ventsim’, ‘VnetPC’ and ‘Vuma’. ‘Ventsim’, ‘VnetPC’ and ‘Vuma’ were not designed to handle fire effects on mine networks. Under the project a small subroutine has been written to transfer the input data from the existing mine ventilation network simulation programs to ‘Ventgraph’.

It is difficult to predict the pressure imbalance and leakage created by a mine fire due to the complex interrelationships between the mine ventilation system and a mine fire situation. Depending on the rate and direction of dip of the entries (dip or rise), reversal or recirculation of the airflow could occur because of convection currents (buoyancy effect) and constrictions (throttling effect) caused by the fire. This reversal jeopardises the functioning of the ventilation system. Stability of the ventilation system is critical for maintaining escapeways free from contamination and therefore available for travel. Reversal of air following fires can have a tragic outcome (Wala, 1999).

Simulation software has the great advantage that underground mine fire scenarios can be analysed and visualised. A number of fire simulation packages have been developed to allow numerical modelling of mine fires (such as Greuer, 1984; Stefanov et al., 1984; Deliac, Chorosz and D’Albrand, 1985, Greuer, 1988; Dziurzyński, Tracz and Trutwin, 1988). The Ventgraph fire simulation program has been described in detail by Trutwin, Dziurzyński and Tracz, 1992. The software provides a dynamic representation of a fire’s progress in real time and utilises a colour-graphic visualisation of the spread of combustion products, $O_2$ and temperature throughout the ventilation system. During the simulation session the user can interact with the ventilation system (eg hang brattice or check curtains, breach stoppings, introduce inert gases and change fan characteristics). These changes can be simulated quickly allowing for the testing of various fire control and suppression strategies. Validation studies on Ventgraph have been performed using data gathered from a real mine fire as undertaken by Wala et al, 1995.

The primary objective of the part of the study described in this paper is to use mine fire simulation software to gain better understanding of how spontaneous combustion initiated fires can interact with the complex ventilation behaviour underground during a substantial fire. It focuses on the simulation of spontaneous combustion sourced heatings that develop into open fires. It examines ventilation behaviour effects of spontaneous combustion initiated pillar fires. It examines the difficulties these can be present if a ventilation reversal occurs. It also briefly examines simulation of use of the inertisation to assist in mine recovery.

EFFECTS OF FIRES ON MINE VENTILATION
The effects of fire on a mine ventilation network are complex. An open fire causes a sharp increase in the temperature of the air. The resulting expansion of the air produces a number of distinct effects. First the expansion attempts to take place in both

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directions along the airway. The tendency to expand against the prevailing direction produces a reduction in the airflow. Secondly, the expansion in volume increases air velocity downwind from the fire causing additional pressure loss. This is known as the choke or throttling effect. Finally, the decreased density results in the heated air becoming more buoyant and causes local effects as well as changes in the magnitudes of natural ventilating energy.

The choke or throttling effect
This effect results from an increase in volume of air as it passes through the fire. The effect has been described by Litton et al., 1987. This increase in volume is due to gas expansion as well as the addition of combustion products such as fire gases and evaporated water. As a result the velocity of air downwind from the fire is increasing and additional pressure loss following the square law results.

The choke effect is analogous to increasing the resistance of the airway. For the purposes of ventilation network analyses based on a standard value of air density, the raised value of this ‘pseudo resistance’, \( R_t \), can be estimated in terms of the air temperature as follows (McPherson, 1993).

\[ R_t \propto T^2 \]

The value of \( R_t \) increases with the square of the absolute temperature (\( T \)). However, it should be recalled that this somewhat artificial device is required only to represent the choke effect in an incompressible flow analysis.

Buoyancy (natural draft) effects

Local or roll back effect
The most immediate effect of heat on the ventilating air stream is a very local one. The reduced density causes the mixture of hot air and products of combustion to rise and flow preferentially along the roof of the airway. The pronounced buoyancy effect causes smoke and hot gases to form a layer along the roof and, under low air velocity in a level or descentional airway, may back up against the direction of airflow. This has been discussed by Mitchell, 1990.

Whole mine natural ventilation pressure effects
A more widespread effect of reductions in air density is the influence felt in shafts or inclined airways. The conversion of heat into mechanical energy in the ventilation system is called the buoyancy (natural draft, natural ventilating pressure or chimney) effect. The effect is most pronounced when the fire itself is in a shaft or inclined airway and promoting airflow if the ventilation is ascentional and opposing the flow in descentional airways. In the ascentional situation flows can reverse in parallel (bypass) airways to the airway with fire and bring combustion products into these airways. In the descentional case airflow may reverse in the airway with fire, bringing combustion products into adjacent parallel airways and also resulting in non-steady state flow of toxic atmospheres.

Natural ventilating pressure always exists in a mine and its magnitude mostly depends on the mine’s depth and difference in air density in the inclined and vertical airways. In the case of fire, this effect is magnified due to high temperatures leading to unpredictable changes in air density and the airflow distribution.

If the air temperatures can be estimated for paths downstream of the fire then it is possible to determine the modified natural ventilating pressures. Those temperatures vary with respect to size and intensity of the fire, distance from the fire, time, leakage of cool air into the airways affected and heat transfer characteristics between the air and the surrounding strata.

ANATOMY OF A HEATING
The development of a spontaneous heating in coal is a complex phenomenon and is poorly understood. This is at least in part because it is so difficult to observe real heatings and particularly those in underground coal mines which have the greatest potential to cause damage. Much of what is known about spontaneous heatings is derived from laboratory studies of very small scale tests. These tests provide valuable data on the many parameters that affect the oxidation process and the production of off-gases that might be used for detection purposes. It is, however, virtually impossible to comprehensively examine a real heating, to measure its temperature distribution, to measure the airflow involved, or to measure almost any aspect of its behaviour. It is impossible to conduct an ‘autopsy’ on a heating to see what has happened. Considerable insight can, however, be gained into the nature of real heatings by examining in detail the results obtained from a simulated heating.

All spontaneous heatings require that certain conditions are satisfied for the coal temperature to continue to rise. Primary amongst these conditions is that, at some point within the pile or solid mass of coal, the rate of heat generation from oxidation exceeds the rate of heat loss due to conduction and convection. If ever this condition is not fulfilled, the heating will have reached a maximum temperature and there will be no further increase. The temperature in the pile or mass of coal will henceforth begin to decrease. Whilst the requirement for this condition is well known, it is difficult to predict the characteristics of coal mass or size, coal reactivity, airflow flux and other parameters that will allow the development of a high temperature heating.

Model used to examine the development of a spontaneous heating
Humphreys, 2004 has developed a numerical model to examine the development of a heating within a coal pillar or pile. For the purposes of modelling, it has been assumed that the starting conditions in the coal pile are homogenous; that is with all coal at the same particle size, reactivity and initial temperature. An underground coal pillar or solid mass will have a permeability that allows passage of air as controlled by the mine ventilation air pressure across the pillar. This permeability is likely to be lower than that exhibited by loose coal in a pile although the spontaneous combustion development characteristics will follow the same trend. In Humphreys’ analysis the airflow flux is constant across the model although obviously there is consumption of oxygen as air passes through the model. For the purposes of examining the nature of a spontaneous heating as it occurs in a pile of coal, a quasi three-dimensional model has been run for a representative coal.

Humphreys, 2004 summarises the base case modelling parameters in Table 1.

The output from the model was selected to give the distribution of coal temperature and oxygen content in the pile during the development of the heating. This data has been used to show the development of a heating to the point that would represent a significant hazard if present in a coal mine or stockpile.

Development of a spontaneous heating
At the commencement of the simulated heating, the temperature throughout the pile was set at 35°C and it was noted that there was linear gradation in oxygen content across the pile from the upwind surface at 21 per cent toward the downwind surface where it had fallen to 17 per cent. Some aspects of this development are illustrated in Figure 1. During the very early stages of the heating development, the oxygen concentration in the pile tends to rise as the initial high rate of oxidation diminishes, due to the effect of accumulated oxidation. The
Oxygen concentration is quickly dominated by the effect of increased temperature in the centre of the pile. A temperature gradient is also rapidly established around in the pile as the interior begins to heat but the temperature at the edges remains low due to heat losses to the surroundings.

The simulation model shows that at about 444 hours the peak pile temperature has reached 55°C about 1875 mm from the upwind surface and the temperature distribution already shows a steeper gradient at the front of the pile than at the back. The oxygen concentration is everywhere more than 17.8 per cent, although there is a gradient from the front to the back of the pile. The rate of oxidation is therefore not significantly affected by the oxygen content at any point in the pile, but will be slightly higher at the front of the pile. The minimum oxygen concentration occurs at the back surface on the centreline of the pile. The rate of self-heating is only about 0.1°C/hour.

By 574 hours, the peak temperature has reached 95°C still at about 1875 mm from the upwind surface of the pile and the minimum oxygen content has fallen to nine per cent at the downwind surface of the pile. At the position of the highest temperature, the oxygen concentration is still in excess of 15 per cent. This will not significantly limit the rate of self-heating which has reached about 1.25°C/hour. This is a considerable increase over the rate at 75°C and it can be expected to continue to increase. There appears to be a distinct ‘hotspot’ forming with only about six per cent of the coal hotter than 75°C.

The hotspot becomes more significant as the heating continues to develop. By 585 hours, it has reached 119°C and has moved slightly forward in the pile to about 1825 mm. The rate of self-heating in the pile continues to rise and has reached about 2.6°C/hour. There is a significant gradient in the oxygen concentration from the front to the back of the pile, where the minimum oxygen concentration has fallen to less than three per cent. The most significant reduction in the oxygen content is on the centreline of the pile in the region of the hotspot. This gives rise to a complicated mass/temperature/oxygen distribution in the pile whereby, in some parts of the pile, the rate of oxidation is now significantly reduced due to the low oxygen concentration irrespective of the coal temperature. The oxygen concentration at the hotspot however, remains above 12 per cent and therefore the oxidation rate and self-heating rate are not significantly affected by any oxygen reduction. At the hotspot, the rate of self-heating is about 75 per cent of that in air. The areas most affected by oxygen reductions at this stage are those downwind of the hotspot.

Over the next few hours, there is a significant change in the nature of the heating. The temperature of the hotspot continues to rise and the temperature profile upward of the hotspot becomes such that all the oxygen entering the pile on the centreline is consumed before reaching the hotspot. Because the rate of oxidation at the hotspot must also have fallen to zero due to the low oxygen concentration, the temperature at that point can no longer rise through oxidation, but only by heat transfer to it from other ‘hotter’ parts of the pile. The only points in the pile that can become hotter than the current hotspot are those upwind on the centreline, where oxygen is still available. All points downwind are receiving no oxygen. All other points off the centreline will have higher heat losses through conduction to ‘cooler’ parts of the pile. This causes the hotspot to migrate forward in the pile.
By 610 hours, the peak temperature has reached 277°C and the hotspot has migrated forward to be 1075 mm from the upwind surface of the pile. This coincides with the position of the minimum oxygen concentration in the pile which has fallen to zero per cent.

The rate of self-heating in the pile tends to decrease for a period between 620 and 680 hours. The peak temperature in the pile remains fairly constant at about 340°C but the position of the hotspot migrates forward from 675 mm to 275 mm leaving behind substantial quantities of hot coal which can no longer oxidise due to the low oxygen levels. This is evident in changes in the mass/temperature distributions and there is a significant increase in the coal mass above 300°C over this period.

There is an obvious limit to the forward movement of the hotspot when it encounters the upwind surface of the pile at about 675 hours. This triggers another change in the behaviour of the heating and there is another rapid rise in the peak coal temperature. At 685 hours, the peak temperature has reached 467°C and is located at the upwind surface of the pile. At this point, it could be expected that the heating will cause the outbreak of open fire, if the coal is loose enough or falls away from the side of the pile, and excess oxygen becomes available to the hot coal. This transition is likely to be very rapid as the temperature gradients ahead of the hotspot are very high, exceeding 2000°C/metre during this phase of the heating.

Alternately, it is possible that the heating continues without fire at the upwind surface and the peak temperature continues to climb and eventually leads to the formation of a charline, when the heating consumes all the reactive portion of the coal. For this heating, this occurs after 705 hours and the coal has reached a peak temperature of about 570°C. The cumulative loss of reactive material is such that the coal can be regarded as being coked and a charline is formed in the pile. The position of the charline is superimposed upon the oxygen distribution at 710 hours in Figure 2.

Upwind of the charred zone, oxidation and pyrolysis has consumed the entire reactive portion of the coal leaving an unreactive coke or char material. In the mass-temperature distribution, the mass of coal at a particular temperature range is split between oxidising and unoxidising. Unoxidising coal represents that part of the original coal lost to oxidation and pyrolysis. Much of this is in the form of solid char, although some is lost as gaseous products of oxidation and pyrolysis.

There is a very significant increase in the unoxidising coal once the temperature exceeds about 550°C and char is formed in the pile as shown.

The heating has now entered its final phase of charring and, unless halted by some other process, continues to char the whole pile as illustrated in Figures 1 and 2 between 700 and 800 hours. The charline expands laterally and migrates downwind leaving behind hot unreactive coke.

In summary the development of this heating can be summarised in the two fairly simple Figures 1 and 2 showing the important features discussed above. The peak temperature in the pile, the position of the peak temperature (the hotspot) and the position of the minimum oxygen concentration in the pile are shown in Figure 1. The development of the charline is shown in Figure 2. The main features that have been discussed above are readily visible. At the very start of the heating, there is a moderately rapid increase in temperature, with the ‘hotspot’ located at the upwind surface of the pile and the minimum oxygen concentration at the back surface. The rate of temperature rise moderates (not visible on the figure but occurs nevertheless) and the position of the peak temperature moves gradually downwind. The position of the minimum oxygen remains at the back surface of the pile, although the minimum oxygen concentration is decreasing. After 275 hours, the peak temperature has moved to the furthest downwind position at about 1900 mm. The hotspot remains in this position until its temperature exceeds 125°C.

This triggers a change in the behaviour of the heating and the hotspot begins to migrate forward. Shortly afterwards, the minimum oxygen concentration in the pile falls to zero, as does the oxygen concentration at the hotspot. Despite this, the peak pile temperature is increasing rapidly, at approximately 8°C/hour.

Once the positions of the peak temperature and minimum oxygen concentration coincide, they begin to migrate together toward the upwind surface. This can only begin when the temperature profile in the coal ahead of the hotspot is sufficient to consume all the oxygen entering that part of the pile. The forward migration of the heating is limited by the upwind surface which triggers another increase in the coal temperature. A short while after this, the temperature of the coal is sufficient to cause charring and a charline is formed in the pile. The final phase of the heating is the lateral expansion and downwind migration of the charline, as all the reactive elements in the coal are consumed by oxidation.

![Figure 2 - Base case heating development – charline development.](image-url)
From this analysis, it is possible to divide the development of this heating into three distinct phases:

1. The incipient phase characterised by peak temperatures up to about 125°C. During this phase a hotspot develops from the upwind surface, migrates downwind to a maximum depth and remains static in that position.

2. The migration phase characterised by the forward migration of the hotspot. During this phase the oxygen concentration falls to zero per cent and there is a very rapid increase in the peak coal temperature. Without remedial action, the heating continues to develop and could lead to the outbreak of fire at the upwind surface of the pile.

3. The charring phase, when the temperature in the pile is sufficient to cause the formation of unreactive char. Without remedial action the heating will continue to develop until the hotspot and charline encounter the downwind surface when an open fire could break out.

These phases have been shown in Figures 1 and 2. The most significant phase in any heating is the incipient phase to about 125°C. Any spontaneous heating which is sufficiently large as to pose a threat to safety will have to pass through the incipient phase. Most of the time required for a dangerous heating to develop will be in reaching 125°C. There may be circumstances in which a coal can be exposed to airflow and such a heating will not develop. For example, if the mass or thickness of the coal pile is insufficient, heat losses will predominate at some temperature and a spontaneous heating will not occur. However, for heatings of significance, the incipient phase time period will be significant and there are a number of important factors in determining whether spontaneous combustion will occur in a particular coal.

SIMULATION OF A SPONTANEOUS COMBUSTION INDUCED MINE FIRE

Ventgraph fire simulation software has been used to examine and illustrate the effects on the mine ventilation network of an open fire on a pillar sidewall rib induced by a spontaneous combustion heating developed from within the pillar. The simulation illustrates the effects of the fire on the whole mine ventilation network after an incubation period of about 700 hours following the outbreak of the pillar fire following a long incipient period and a migration phase upwind. The pillar under examination is positioned separating a mains intake heading from a return pillar; and

1. an open fire that has broken out on the intake side of the pillar; and

2. a subsequent stage when an open fire has broken out on the return side of the pillar (the charring phase, when the heating has continued to develop until the hotspot and charline encounter the pillar rib on the return side).

This hypothetical spontaneous combustion incident is reported as a simulation scenario that focuses on effects across the whole mine network. It is written up as a series of developments against time from the outbreak of the open fire in the pillar rib.

FIRE SCENARIO DEVELOPMENT

Spontaneous combustion fire in fractured pillar coal in the rib of F Heading inbye 27 c/t. There is a very high pressure of about 1200 Pa across the F to G (intake/return) pillar. Heating started as deep-seated oxidation. In the initial stages of heating, moisture transfer and coal oxidation predominate. Mains entry nomenclature is as follows:

- Headings C and D are intake transport roads,
- Heading E is the intake belt road,
- Heading F is another intake road (second means of egress), and
- Headings B, G, H and I are returns.

Intake side fire following the migratory phase

As the coal dries out, a substantial local hot spot develops near the air inlet and begins to migrate upwind. Heating front has moved upstream in search of oxygen to the F Heading pillar rib. It has just developed to the point of an open fire. Prior to running the Ventgraph simulation mine ventilation and gas characteristics and monitoring controls that may be required are pre-entered.

- CO and CH₄ electronic sensors inbye the fire at 4 N LW and 5N development TG Dog Legs;
- CO sensors in E Heading 10 - 11 c/t and 38 - 30 c/t; and
- CH₄ sources of 0.4 m³/s from 4N LW face and 5N development face.

Assume CO sensors in control room have alarm set at 8 ppm.

Simulation

- **Step 1** – Time 0 - 30 minutes: simulate 1 m length open fire over entry width. Smoke first reaches 5N development face at 22 minutes.

  **Control:** Hypothetical action of development face crew. Crew see smoke and phone outbye deputy at 30 minutes. Crew contact control room operator (CRO) and CRO ask LW crew to evacuate mine. Crews drive out in smoke. Crews reach surface at time since fire outbreak of 45 minutes.

- **Step 2** – Time 30 - 60 minutes: coal fire grows. 5 m entry length coal burning.

  **Control:** Hypothetical action of deputy:
  - deputy finds fire source at 45 minutes after fire start, and
  - deputy has hose ready to fight by 60 minutes.

- **Step 3** – Time 60 - 90 minutes: coal fire grows. 25 m entry length coal burning.

  **Control:** Deputy cannot extinguish fire at 90 minutes and drives out of mine. Reaches surface at 105 minutes. Fire out of control. Decision reached that underground ventilation control will be ineffective. Decision made to shut down the two underground booster fans and one main fan.

- **Step 4** – Time 90 - 120 minutes: continue coal fire. 25 m entry length coal burning.

  No CO sensors in mine have alarmed yet.

- **Step 5** – Time 120 - 180 minutes: coal fire grows. 50 m entry length coal burning.

  CO sensors on 5N development Dog Leg first alarms at 130 minutes.

  Air carrying significant CH₄ never reaches the fire zone during the simulation. CO sensor 4N LW Dog Leg has not alarmed after 180 minutes.

- **Step 6** – Time 180 - 240 minutes: coal fire grows. 100 m entry length coal burning.

  CO sensor 5N Dog Leg is alarming at 240 minutes.

- **Step 7** – Time 240 - 420 minutes: continue coal fire, 100 m entry length coal burning.

  CO sensor in E Heading 32 c/t alarming and smoke has reached LW face.
Fig 3 - Smoke distribution after 22 minutes. Some smoke is reaching surface exhausting main fans outlets.

Fig 4 - CH₄ distribution from mine seam gas after 22 minutes.
FIG 5 - Smoke distribution, heat production and fire temperature after booster and mine fans turned off 97 minutes after fire started.

FIG 6 - CO distribution after 120 minutes.
Fig 7 - CH₄ distribution after 150 minutes.

Fig 8 - CO distribution after 420 minutes.
FIG 9 - Smoke distribution, heat production and fire temperature after 420 minutes.

FIG 10 - Smoke distribution after five minutes.
Fig 11 - CO distribution levels after 150 minutes.

Fig 12 - CO distribution levels, heat production and fire temperature after 360 minutes.
Return side fire following the charring phase

Following the charring phase, when the heating has continued to develop until the hotspot and charline encounter the pillar rib on the return side an open fire has broken out on the downwind side of the pillar.

A spontaneous combustion initiated fire in fractured rib pillar coal in G Heading (return) outbye 27 cut through and near the No 2 booster fan. There are no electronic sensors inbye the fire.

Simulation

• **Step 1** – Time 0 - 360 minutes: simulate 1 m length fire over entry width.
• **Step 2** – Time 360 - 720 minutes: continues coal fire 5 m entry length coal burning.

Both these intake and return side scenario simulation could be undertaken for much longer on the assumption that coal within the mine continues to burn and no remedial action such as flooding or introduction of gas inertisation occurs. It has shown how a relatively common form of mine fire, a spontaneous combustion initiated coal pillar fire (with the pillar separating intake and return air and with substantial pressure differences) can affect the mine workings. It has shown how CO levels in mine airways increase over time for a specific fire build up scenario.

In the intake side fire significant CO levels reach the mains and 5N development faces early but also eventually reach the 4N LW face if the fire is not stabilised and extinguished. The fumes from the fire have only limited effect on the LW face as it receives most of its intake air from mains C and D transport roads.

In the return side fire significant CO levels build up. However, these pollutants are restricted to the return airways and so do not directly imperil miners who are evacuating the mine.

**CONCLUSIONS**

A study has examined the potential for simulation of the effects of a relatively common form of mine fire, a spontaneous combustion initiated coal pillar fire on a mine ventilation network. The project involved applying the ‘Ventgraph’ mine fire simulation software to preplan for mine fires and possible emergency evacuations.

The background to this approach to simulating the effects of mine fires on the mine ventilation network has been examined. The anatomy of a spontaneous combustion heating has been analysed. The three stages in the development of a mine pillar or pile heating, namely the incipient phase, the migration phase characterised by the forward migration of the hotspot and possible open fire on the forward surface and the charring phase, when without remedial action the heating will continue to develop until the charline encounters the downwind surface when an open fire could break out.

A case study of the simulated effects of fumes from a fire on the ventilation of a modern Australian mine has been examined. Mine fires are recognised across the world as a major hazard issue. New approaches allowing improvement in understanding their consequences have been developed as an aid in handling this complex area.

The mine fire simulator Ventgraph has been shown to be an important tool in planning for mine fires developed from spontaneous combustion heatings. The capability to visually...
display the spread of effects of a fire quickly and reliably provides a strong aid to those involved in developing emergency plans or contributing to emergency management. The active use of mine fire simulation in emergency planning should continue to be encouraged.

ACKNOWLEDGEMENTS

The support efforts of the University of Queensland, the Australian Coal Association and a number of operations within the Australian mining industry are acknowledged.

REFERENCES


Humphreys, D, 2004. The application of numerical modelling to the assessment of the potential for, and the detection of, spontaneous combustion in underground coal mines, PhD thesis (unpublished), University of Queensland.


A Review of Spontaneous Combustion Incidents
B Ham

ABSTRACT
Heatings, fires and explosions in New South Wales (NSW) and Queensland coal mines since 1972, are reviewed. Geological and mine settings are outlined. Where available, gas analysis and critical decisions are discussed. This paper updates work by NSW Department of Mineral Resources (1995) and the Queensland Department of Mines and Energy in a report on fires and explosion in Queensland coal mines (Richardson and Ham, 1996).

Incident data has important applications in terms of evidence-based risk assessment and identifying key issues to be integrated into competency based training programs. Predictive indicators are reviewed in terms of coal parameters in seams where the incidents occurred.

INTRODUCTION
Over the last five years there has been an increase in the number of spontaneous combustion incidents, culminating in the closure of Southland Colliery in December 2003. There have been some significant technical advances in risk identification and management of spontaneous. The changing demographics of managers and technical officers and legislated requirements for risk management and adequate training and maintenance of competency programs has created a need to review and analyse spontaneous combustion incidents. Since 1972, spontaneous combustion has caused three underground mine explosions with a total loss of 41 lives in Queensland and several extended and permanent pit closures in New South Wales.

BACKGROUND
The recommendations of the Inquiry into the 1994 Moura Mine disaster hastened the development of improved mining practices through:
1. legislation supporting risk management processes,
2. legislation supporting improved training and maintenance of competency programs,
3. improvements in monitoring,
4. improvements in emergency management,
5. the development of inertisation programs, and
6. improved incident reporting and analysis systems.

Most of these aspects are in continuous improvement processes.

One issue identified in the inquiry was the lack of an incident reporting system that made available information on the frequency and causes of spontaneous combustion (and other low frequency but high impact) incidents. To inform the Inspectorate and industry more generally, the Department commissioned the Mining Engineer from the Queensland Coal Board and a research officer from SIMTARS to review the monthly Inspectors reports from 1962 to 1995 and to extract incident data (Richardson and Ham, 1996). From 1996 to 1998 the Inspectorate developed an incident reporting system that was launched in 1999.

From about 1989, the New South Wales Inspectorate has been compiling and reporting on incident data as published by the Department of Mineral Resources (1995).

The technology to monitor spontaneous combustion through continuous analysis of mines gases was developed by SIMTARS after the 1986 Moura Explosion and this process is continuously being refined Cliff et al (1994), Cliff (1995) and Cliff, Rowlands and Sleeman (1996).

The conventional wisdom of dealing with spontaneous combustion in mining panels was to extract the mining equipment and seal the area. The recent development of inertisation procedures to treat and manage developing heatings is a milestone in the prevention and control of spontaneous combustion in retreat mining (Task Group 5 [Moura Inquiry Implementation] Report, 1997). This approach enables confident mining using productive longwall technology that does not have the flexibility of the previous continuous miner systems (Humphries, 1999; Blanche and Stephon, 2000). The failures and successes reported by these authors has been a starting point for a revival in application of new technologies for spontaneous combustion management and associated training. Recent discussion forums include programs such as the Mackay Inertisation Seminar and presentations at the Colliery Managers Association Meeting in Belmont New South Wales.

REPORTING OF SPONTANEOUS COMBUSTION INCIDENTS
The standards of reporting and presenting spontaneous combustion incident data are slowly evolving but arguably not yet at a pace to meet the industry’s needs. The legislation requires managers and engineers to effectively manage spontaneous combustion risk at mines and to maintain an adequate level of skill. The New South Wales Department of Mineral Resources (1996) published some useful guidance for spontaneous incident reporting in the NSW Spontaneous Combustion Management Code. The issues recommended to be included in spontaneous combustion incident reports are shown in Table 1.

<table>
<thead>
<tr>
<th>TABLE 1</th>
<th>Recommended spontaneous combustion incident data set – NSW DMR.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine</td>
<td>Description of event</td>
</tr>
<tr>
<td>Seam(s)</td>
<td>Method of control</td>
</tr>
<tr>
<td>Proximate analysis</td>
<td>Ventilation details</td>
</tr>
<tr>
<td>Area and depth of origin</td>
<td>Mining systems issues</td>
</tr>
<tr>
<td>Location of event</td>
<td>Panel design issues</td>
</tr>
<tr>
<td>Mine conditions</td>
<td>Ventilation issues</td>
</tr>
<tr>
<td>Environmental conditions</td>
<td>Pressure differential issues</td>
</tr>
<tr>
<td>Exposure time</td>
<td>Airways and appliances</td>
</tr>
</tbody>
</table>

Research work on spontaneous combustion incident reporting funded by ACARP (Cliff, Mobrey and Meadowscroft, 2003) helps fill this gap, but the issues of ownership, compilation, analysis and reporting of the data set still have to be addressed.

At present, the majority of available data is extracted from Inspectors monthly reports. While these reports were not prepared for educational or risk management purposes, collating

1. MAusIMM, Consulting Mining Engineer and Health and Safety Advisor, 39 Victoria Crescent, Toowong Qld 4066. Email: bruceham@optusnet.com.au
and analysing the data provides some interesting perspectives on
the nature of spontaneous combustion risk and how it has been
assessed and managed.

Richardson and Ham (1996) reported on spontaneous
combustion references in Queensland coal mining inspectors'
reports. From 1972 to 1994, there were 39 reports of spontaneous
combustion incidents in underground coal mines. A review of the
literature has revealed a further eight incidents at mines
including Newlands, Blair Athol and North Goonyella between

OCCURRENCES OF SPONTANEOUS COMBUSTION INCIDENTS

A total of 51 spontaneous combustion incidents have been
identified in Queensland between 1972 and 2004. These are
listed in Appendix 1. The most devastating of these were Box
These incidents resulted in explosions that resulted in 17, 13 and
11 fatalities respectively as well as closures of the mines.

Most of the incidents (35) were located in the Bowen Basin as
shown in Table 2. The pattern of mine closures in the West
Moreton Field is reflected in the reduction of incidents, with only
two events in the last 15 years.

### Table 2

<table>
<thead>
<tr>
<th>District</th>
<th>Heatings</th>
</tr>
</thead>
<tbody>
<tr>
<td>West Moreton</td>
<td>15</td>
</tr>
<tr>
<td>Bowen Basin</td>
<td>35</td>
</tr>
<tr>
<td>Newcastle and Hunter</td>
<td>36</td>
</tr>
<tr>
<td>Other</td>
<td>2</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>88</strong></td>
</tr>
</tbody>
</table>

In classifying the mine-site location of incidents, the
comments in the inspectors reports did not always provide
adequate information particularly in relation to pillar extraction
and pillar splitting. Using personal knowledge of operations and
engineering judgements based on pillar stability issues,
spontaneous combustion incidents have be subdivided by mine
location as shown in Table 3. Most incidents occur in pillar
splitting followed by pillar extraction. Recently, with the
introduction of longwall operations in thicker seams and more
active seams, a number of incidents have occurred in association
with longwall mining. A few incidents have occurred in the
pillars in main entries where the pressure differential between
intakes and returns has caused spontaneous combustion in
fissures in pillars.

### Table 3

<table>
<thead>
<tr>
<th>Location</th>
<th>Incidents</th>
</tr>
</thead>
<tbody>
<tr>
<td>Main Entries</td>
<td>8</td>
</tr>
<tr>
<td>Pillar splitting</td>
<td>18</td>
</tr>
<tr>
<td>Pillar Extraction</td>
<td>11</td>
</tr>
<tr>
<td>Longwall</td>
<td>6</td>
</tr>
<tr>
<td>Other</td>
<td>8</td>
</tr>
</tbody>
</table>

The other category includes spontaneous combustion
associated with piles of broken coal, in open cut mines and also
in re-opened old workings.

At first there appears to be a decrease in incidents in time as
shown in Table 4, but when the effect of the closure of
underground mines in the West Moreton is taken into account,
the incidence of spontaneous combustion events is relatively
constant over time.

### Table 4

<table>
<thead>
<tr>
<th>Year</th>
<th>Incidents Queensland</th>
</tr>
</thead>
<tbody>
<tr>
<td>1971 - 1975</td>
<td>9</td>
</tr>
<tr>
<td>1976 - 1980</td>
<td>14</td>
</tr>
<tr>
<td>1981 - 1985</td>
<td>5</td>
</tr>
<tr>
<td>1986 - 1990</td>
<td>7</td>
</tr>
<tr>
<td>1991 - 1995</td>
<td>6</td>
</tr>
<tr>
<td>1996 - 2000</td>
<td>5</td>
</tr>
<tr>
<td>2001 - 2004</td>
<td>5</td>
</tr>
</tbody>
</table>

IMPACTS OF SPONTANEOUS COMBUSTION INCIDENTS

The examination of incidents is a useful part of the risk
assessment process in that it provides some evidence of the
adverse impacts of spontaneous combustion, which include:

- fatalities through blast trauma and asphyxiation from CO
  poisoning,
- mental disorders in survivors of disaster,
- mine closures,
- loss of equipment,
- loss of production,
- loss of reputation and market position,
- costs of distraction to management,
- costs of remedial and recovery measures,
- cost of disruption to communities,
- costs of preventative measures,
- costs of monitoring systems,
- costs of developing a spontaneous combustion management
  system, and
- costs of associated training.

SPONTANEOUS COMBUSTION RISK FACTORS
AND CONTROLS

The examination of incidents is a useful part of the risk control
process in that it provides some evidence of the successes and
failures in assessing and controlling risk. For example:

- analysis of coal type and self-heating rates $R_{70}$,
- review of mining methods,
- review of ventilation, control systems and management,
- review of atmospheric monitoring,
- review of inspection programs, and
- review of training.

Work in examining spontaneous combustion risk from a coal
geology perspective was pioneered in Australia by Humphreys,
Rowland and Cudmore (1981) and more recently developed by
Beamish, Barakat and St George (2000 and 2001). The method
used involves measuring the time a crushed coal sample takes to
rise from 40°C to 70°C. This is called the self heating rate $R_{70}$.
and is shown to be a function of coal rank (Beamish, in press) but is affected by ash (Beamish and Blazak, in press) and moisture content (Beamish and Hamilton, in press).

Using coal quality data from Department Natural Resources and Mines (2001) and Edwards (1975), the atomic hydrogen/carbon and oxygen/carbon rations and be calculated to estimate the Suggate Rank after Beamish (in press). This also facilitates an estimation of the self-heating rate $R_{70}$. Table 5 shows the estimated $R_{70}$ values compared with actual results from various ACIRL tests in the 1980s.

### Table 5

<table>
<thead>
<tr>
<th>Seam/State</th>
<th>Atomic H/C</th>
<th>Atomic O/C</th>
<th>Suggate rank</th>
<th>Self-heating rate $R_{70}$</th>
<th>Estimate</th>
<th>Test result</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Queensland</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Bowen</td>
<td>0.67</td>
<td>0.04</td>
<td>13</td>
<td>0.05</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Blake</td>
<td>0.65</td>
<td>0.06</td>
<td>14</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Castor/Argo</td>
<td>0.68</td>
<td>0.04</td>
<td>13</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>Peak Downs</td>
<td>0.66</td>
<td>0.03</td>
<td>15</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Castor/Argo</td>
<td>0.68</td>
<td>0.05</td>
<td>14</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>Moura A, B, C &amp; D</td>
<td>0.75</td>
<td>0.07</td>
<td>12</td>
<td>1</td>
<td>0.4</td>
<td>0.4</td>
</tr>
<tr>
<td>Amberley Series</td>
<td>0.97</td>
<td>0.11</td>
<td>9</td>
<td>9</td>
<td>1.5</td>
<td>1.5</td>
</tr>
<tr>
<td>Ipswich CM</td>
<td>0.89</td>
<td>0.11</td>
<td>9</td>
<td>9</td>
<td>1.2</td>
<td>1.2</td>
</tr>
<tr>
<td>Blair Athol</td>
<td>0.67</td>
<td>0.10</td>
<td>11</td>
<td>2</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>Newlands Lower</td>
<td>0.71</td>
<td>0.07</td>
<td>12</td>
<td>1</td>
<td>0.4</td>
<td>0.4</td>
</tr>
<tr>
<td>Middle Goonyella</td>
<td>0.72</td>
<td>0.04</td>
<td>14</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td><strong>New South Wales</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dudley</td>
<td>0.80</td>
<td>0.07</td>
<td>12</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Greta</td>
<td>0.85</td>
<td>0.08</td>
<td>11</td>
<td>1.5</td>
<td>1.5</td>
<td>1.5</td>
</tr>
<tr>
<td>Great Northern</td>
<td>0.73</td>
<td>0.09</td>
<td>12</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Ulan</td>
<td>0.70</td>
<td>0.08</td>
<td>12</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
</tbody>
</table>

Beamish and Daly (2004) report on the increased risk of a heating when gas drainage is applied. Gas drainage dries the coal and increases available reactive surface area and decreases the heat adsorbed in removing moisture. In laboratory tests, the time for a dried sample to heat (25°C to 200°C) decreased from 22 days to seven days.

### RISK MANAGEMENT PROCESSES

Given the low frequency and more importantly the severity of recent incidents, particularly in New South Wales, it is necessary to review the risk management processes and understand why the risk management systems have failed (Brady, 2004). The legislation in both Queensland and New South Wales requires mine operators to adopt a risk management process for control of major hazards such as spontaneous combustion incidents and particularly related mine fires and explosions.

The Department of Mineral Resources (1996, 1997a, 1997b, and 2000) in New South Wales has published numerous documents directly or indirectly related to spontaneous combustion. These include spontaneous combustion management code MDG 1006 and a detailed guide to risk assessment processes MDG 1010. While useful, these guides provide the framework and not the detail for spontaneous combustion risk management.

### APPLICATIONS TO TRAINING

The analysis of incidents provides an evidence base for assessing whether there are flaws in the spontaneous combustion risk management processes and how improvements may be made in relation to training and maintenance of competency.

The current focus of the Simtars’ spontaneous combustion training packages (Cliff, Rowland and Sleeman, 1996, revised 2004) is on knowledge of spontaneous combustion. The current trend in general training programs established by the Australian National Training Authority (2004) and the National Mining Industry Training Advisory Body (2004) is to present necessary knowledge, but then ensure that the candidates have the necessary skills to effectively apply that knowledge. When the skill is demonstrated, achievement of competence is acknowledged. Spontaneous combustion management is the focus of the following units:

- MNC.U101.A Apply spontaneous combustion management measures
- MNC.U102.A Establish spontaneous combustion management plan
- MNC.U103.A Implement spontaneous combustion management plan
- MNC.U104.A Apply spontaneous combustion management plan

There is also a spontaneous combustion component at the establish, implement, apply and monitor levels of units that cover:

1. ventilation management plans,
2. manage, operate and maintain the mine ventilation system,
3. gas management plan,
4. mining method and strata management systems, and
5. emergency preparedness management systems.

The application of new technologies demands new skills to be developed in the workforce. The inflexibility of longwall mining has increased the potential losses from a spontaneous combustion incident, but the development of inertisation technology appears to offer an effective control if preparation, monitoring and control actions are well executed. These new skills need to be formally developed and training implemented within the Coal Training Package and at sites where the risk needs to be controlled.

The rapid change in ventilation and gas monitoring options associated with the introduction of inertisation options is providing a significant challenge for professionals to maintain their competency.

### CONCLUSIONS

Spontaneous combustions incidents are low frequency but high impact events that can affect most underground and some surface mines. The effective management of spontaneous combustion incidents relies on sophisticated technology and rapid decision making by a skilled and informed workforce.

The legislation requires mine operators to have hazard management plans for spontaneous combustion. These plans should ensure effective monitoring systems and management procedures are in place. These management systems include a training component that needs to be regularly updated as staff turn-over and technology advances.

The understanding and analysis of spontaneous combustion incidents does not in itself prevent future events, but this analysis needs to be used to develop skills in identifying and managing risks. The analysis of incidents also may be used as a tool to test, audit and develop safety management systems.
Geotechnical assessment of spontaneous risk is a useful tool, but practical experience shows that even when a low risk is shown by either a high Suggate rank or a low self-heating rate, a residual risk remains. Geological features such as roof coal, floor coal and high pyrites are associated with increased risk. Testing needs to consider separate bands of coal. A small but reactive band might be all that is required to initiate a heating.

In the worst spontaneous combustion related disasters, common factors include poor mine planning, a reluctance to acknowledge the seriousness of the situation, inadequate information and inadequate training of key decision makers.

REFERENCES


Coal Services Annual Report, 2002-03, p 12.


Department of Mineral Resources (NSW), 1996. Spontaneous Combustion Management Code Revision 3.3, MDG 1006, Coal Mining and Engineering Branch.


Department of Mineral Resources (NSW), 1997b. Guide to Reviewing a Risk Assessment for Mining Equipment and Operations, MDG 1014.


## A REVIEW OF SPONTANEOUS COMBUSTION INCIDENTS

### APPENDIX 1

Queensland spontaneous combustion incidents.

<table>
<thead>
<tr>
<th>Date</th>
<th>Mine</th>
<th>Worksite</th>
<th>Description</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>31/7/1972</td>
<td>Box Flat No 8</td>
<td>Extraction panel</td>
<td>Explosion occurred while sealing a fire – 17 fatalities</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1977</td>
<td>Box Flat No 8</td>
<td>Goaf</td>
<td>Heating in fallen goaf material</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/3/1977</td>
<td>Box Flat No 8</td>
<td>Panel returns</td>
<td>Test revealed CO, hydrogen and ethylene</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/6/1977</td>
<td>Box Flat No 8</td>
<td>Panel returns</td>
<td>Completed panel sealed after evidence of heating</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>3/6/1983</td>
<td>Box Flat No 8</td>
<td>Main returns/goaf area</td>
<td>Detect smoke and 45 ppm CO, withdraw men, seal entries</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/4/1989</td>
<td>Collinsville No 2</td>
<td>Return 32 C section</td>
<td>Measure 17 ppm CO and seal panel</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/12/1986</td>
<td>Collinsville No 2</td>
<td>Pillar extraction section</td>
<td>Measure 16 ppm CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/7/1993</td>
<td>Collinsville No 2</td>
<td>Pillar extraction section</td>
<td>Measure CO and seal area</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>8/10/1981</td>
<td>Collinsville No 2</td>
<td>Pillar heating</td>
<td>Measure 30 ppm CO in Draeger Tuber</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/3/1980</td>
<td>Collinsville No 2</td>
<td>Pillar extraction – Panel 56 Level</td>
<td>CO measured</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1981</td>
<td>Collinsville No 2</td>
<td>Pillar extraction</td>
<td>Sealed when heating detected</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/9/1984</td>
<td>Collinsville No 2</td>
<td>Goaf area</td>
<td>Sealed after rising CO levels</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/2/1977</td>
<td>Collinsville No 3</td>
<td>Goaf area – Western section</td>
<td>Heating detected</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>19/10/1978</td>
<td>Collinsville No 3</td>
<td>Goaf area – Eastern section</td>
<td>Sealed after heating detected – 98 ppm CO behind seal</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1977</td>
<td>Collinsville No 3</td>
<td>Crushed seal</td>
<td>Ventilation leak causes rising CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1977</td>
<td>Collinsville No 3</td>
<td>Pillar extraction</td>
<td>Heating occurred eight weeks after commencing, area sealed</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/11/1988</td>
<td>Cook</td>
<td>Goaf area</td>
<td>Measure 100 ppm CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/4/1974</td>
<td>Dacon No 3</td>
<td>Goaf area</td>
<td>Heating sealed</td>
<td>Richardson and Ham, 1996</td>
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<tr>
<td>1/10/1991</td>
<td>Harrow Creek</td>
<td>Abandoned workings</td>
<td>Suspect fire reported</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>20/9/1975</td>
<td>Kianga No 1</td>
<td>Underground bord and pillar workings</td>
<td>Explosion – 13 fatalities</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1974</td>
<td>Laleham No 1</td>
<td>Main returns</td>
<td>Heating in pillars between main intakes and returns</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1975</td>
<td>Laleham No 1</td>
<td>Main returns</td>
<td>Heating in pillars between main intakes and returns</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1976</td>
<td>Laleham No 1</td>
<td>Main returns</td>
<td>Heating in pillars between main intakes and returns</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1977</td>
<td>Laleham No 1</td>
<td>Main returns</td>
<td>Heating in pillars between main intakes and returns</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/6/1982</td>
<td>Laleham No 1</td>
<td>Pillar near main tunnel</td>
<td>Smoke detected and area sealed</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/1/1982</td>
<td>Leichardt Colliery</td>
<td>Heap of shot coal</td>
<td>Smoke detected</td>
<td>Richardson and Ham, 1996</td>
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<tr>
<td>1973</td>
<td>Moura No 1</td>
<td>Panel</td>
<td>Heating reported</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1975</td>
<td>Moura No 1</td>
<td>Panel</td>
<td>Heating reported – panel sealed</td>
<td>Richardson and Ham, 1996</td>
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<tr>
<td>1/4/1986</td>
<td>Moura No 2</td>
<td>Panel goaf</td>
<td>High CO reported and panel sealed</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/8/1994</td>
<td>Moura No 2</td>
<td>Extracted panel</td>
<td>Aggressive heating leading to explosion</td>
<td>Richardson and Ham, 1996</td>
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<tr>
<td>1/8/1993</td>
<td>Moura No 4</td>
<td>Entry pillar</td>
<td>Rising CO detected</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/4/1979</td>
<td>New Hope No 5</td>
<td>Panel – 13 West</td>
<td>Measure 55 ppm CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1972</td>
<td>New Witwood No 3</td>
<td>Opencut encountered old workings</td>
<td>Fire developed</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/3/1974</td>
<td>Normanton No 1</td>
<td>Sealed mine</td>
<td>Advanced heating – 6% CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/3/1986</td>
<td>Oakleigh No 3</td>
<td>Retreating panel</td>
<td>After fall 200 ppm CO reported, men with drawn</td>
<td>Richardson and Ham, 1996</td>
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<tr>
<td>1/7/1979</td>
<td>Oakleigh No 3</td>
<td>Over lying old workings</td>
<td>Measured 25 ppm CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/12/1977</td>
<td>Southern Cross No 11</td>
<td>Abandoned panel</td>
<td>Heating reported</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>27/12/1980</td>
<td>Southern Cross No 11</td>
<td>Extracted panel</td>
<td>Sealed on rising CO</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>1/10/1991</td>
<td>Western Leases No 1</td>
<td>Panel extraction – split pillars, extract bottoms</td>
<td>Heating detected</td>
<td>Richardson and Ham, 1996</td>
</tr>
<tr>
<td>5/6/1989</td>
<td>Western Leases No 1</td>
<td>Panel extraction – split pillars, extract bottoms</td>
<td>Heating 89 ppm CO when panel sealed</td>
<td>Spon Commonwealth Course</td>
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<tr>
<td>April 1998</td>
<td>Newlands Underground</td>
<td>Pillar near main portal</td>
<td>Heatings in fractured pillars</td>
<td>Humphries, 1999</td>
</tr>
<tr>
<td>May 1998</td>
<td>Newlands Underground</td>
<td>Pillar near main portal</td>
<td>Heatings in fractured pillars</td>
<td>Humphries, 1999</td>
</tr>
<tr>
<td>28/12/1997</td>
<td>North Goonyella</td>
<td>Longwall 4 south panel</td>
<td>Measure 25 ppm CO in general body and 0.4% hydrogen in goaf area – apply Tomlinson Boiler inertisation</td>
<td>Blanch and Stephan, 2000</td>
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### APPENDIX 1 (continued)

**Queensland spontaneous combustion incidents.**

<table>
<thead>
<tr>
<th>Date</th>
<th>Mine</th>
<th>Worksite</th>
<th>Description</th>
<th>Reference</th>
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<tr>
<td>25/9/1999</td>
<td>North Goonyella</td>
<td>Longwall 5 South panel</td>
<td>Rising CO causes panel to be sealed and inerted</td>
<td>Blanch and Stephan, 2000</td>
</tr>
<tr>
<td>19/3/2003</td>
<td>Withheld C Qld</td>
<td>Longwall</td>
<td>Rising CO in area subject to inertisation, mine evacuated</td>
<td>NRM Incident Database</td>
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<tr>
<td>10/1/2004</td>
<td>Withheld C Qld</td>
<td>Open cut machine</td>
<td>An excavator operator observed smoke. A floor plate was lifted and coal fines were discovered smoldering in the 50 mm gap between the floor plate and the top of the fuel tank</td>
<td>NRM Incident Database</td>
</tr>
<tr>
<td>18/7/2004</td>
<td>Withheld C Qld</td>
<td>Longwall</td>
<td>The Tomlinson Boiler was being used to assist inertisation of SLW1, suffered a functional failure and injected high levels of CO into the goaf area</td>
<td>NRM Incident Database</td>
</tr>
<tr>
<td>18/8/2004</td>
<td>Withheld C Qld</td>
<td>Goaf area</td>
<td>Carbon monoxide was detected behind the seal at a trigger level that required evacuation of the mine (&gt;500 ppm)</td>
<td>NRM Incident Database</td>
</tr>
<tr>
<td>28/8/2004</td>
<td>Withheld Opencut</td>
<td>Fire in pre-strip hole at King 4 (East)5 – Interburden shot heating up where some king seam coal remains</td>
<td>NRM Incident Database</td>
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### APPENDIX 2

**New South Wales spontaneous combustion incidents.**

<table>
<thead>
<tr>
<th>Date</th>
<th>Mine</th>
<th>Worksite</th>
<th>Description</th>
<th>Reference</th>
</tr>
</thead>
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<tr>
<td>13/1/1974</td>
<td>Abedare North</td>
<td>Section entries</td>
<td>Fire in roof coal over stopping, 1000 ppm CO</td>
<td>Lyne and Sneddon, 1981</td>
</tr>
<tr>
<td>Nov 1975</td>
<td>Abedare North</td>
<td>Main pillars</td>
<td>Sealed mine Borehold placed ash used to seal area</td>
<td>MacKenzie-Wood and Ellis, 1981</td>
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<tr>
<td>14/11/1977</td>
<td>Abedare North</td>
<td>Goaf area</td>
<td>Failure of seals</td>
<td>Lyne and Sneddon, 1981</td>
</tr>
<tr>
<td>8/1/1979</td>
<td>West Wallsend No 2</td>
<td>Fire and explosion</td>
<td>standen V</td>
<td>Lampton Colliery</td>
</tr>
<tr>
<td>20/2/1979</td>
<td>Burwood Colliery</td>
<td>Fire</td>
<td></td>
<td>MacKenzie-Wood and Ellis, 1981</td>
</tr>
<tr>
<td>7/12/1979</td>
<td>Lampton Colliery</td>
<td>Goaf</td>
<td>Failure of seals</td>
<td>MacKenzie-Wood and Ellis, 1981</td>
</tr>
<tr>
<td>1989 to 1995</td>
<td>All UG mines = 11</td>
<td>Fire</td>
<td></td>
<td>DMR, 1995</td>
</tr>
<tr>
<td>1989 to 1995</td>
<td>All O/C mines = 6</td>
<td>Fire</td>
<td></td>
<td>DMR, 1995</td>
</tr>
<tr>
<td>1989 to 1995</td>
<td>All prep plants = 8</td>
<td>Fire</td>
<td></td>
<td>DMR, 1995</td>
</tr>
<tr>
<td>1991</td>
<td>Ulan</td>
<td>Longwall</td>
<td>Goaf fire resulted in pit closure for many months</td>
<td>Healey, 1995. DMR Spon Com Seminar Mudgee</td>
</tr>
<tr>
<td>16/5/02</td>
<td>Dartbrook</td>
<td>Longwall goaf fire</td>
<td>Mineshield inertisation at 4 t/min – 10 500 tonnes</td>
<td>Coal Services Annual Report, 2002-03</td>
</tr>
<tr>
<td>15/12/2002</td>
<td>Beltana</td>
<td>Pillar main returns</td>
<td>Pillar heating near fan</td>
<td>Coal Services Annual Report, 2002-03</td>
</tr>
<tr>
<td>1/4/2003</td>
<td>Beltana</td>
<td>Pillar main returns</td>
<td>Pillar fire near outcrop</td>
<td>Coal Services Annual Report, 2002-03</td>
</tr>
<tr>
<td>24/12/2003</td>
<td>Southland</td>
<td>Longwall goaf fire</td>
<td>Mine shield used, mine sealed</td>
<td>Southland Colliery Emergency – Gympie Gold</td>
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<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td><a href="http://www.gympiegold.com.au">http://www.gympiegold.com.au</a></td>
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Computer Animation of Hot Spot Development in Bulk Coal as an Aid for Training Coal Miners

M Hancock¹, M S Kizil² and B B Beamish²

ABSTRACT

The processes that take place during the development of a heating are difficult to visualise. Bulk coal self-heating tests at The University of Queensland (UQ) using a two-metre column are providing graphic evidence of the stages that occur during a heating. Data obtained from these tests, both temperature and corresponding off-gas evolution can be transformed into what is effectively a video-replay of the heating event. This is achieved by loading both sets of data into a newly developed animation package called Hotspot. The resulting animation is ideal for spontaneous combustion training purposes as the viewer can readily identify the different hot spot stages and corresponding off-gas signatures. Colour coding of the coal temperature, as the hot spot forms, highlights its location in the coal pile and shows its ability to migrate upwind. An added benefit of the package is that once a mine has been tested in the UQ two-metre column, there is a permanent record of that particular coal’s performance for mine personnel to view.

INTRODUCTION

Self-heating leading to spontaneous combustion continues to pose a significant hazard during the mining of coal. A recent example of this is Southland Colliery in December 2003, where a heating progressed to ignition forcing the mine to be closed. Unfortunately, the heterogeneous nature of coal and the contributing factors that control whether heat is gained or lost from the coal/oxygen system make it difficult to predict the onset of a heating with any confidence. In addition, it is difficult for mine personnel to visualise how a heating forms. Numerical models appear too complicated for most people and many of these are simplified to enable the complex equations involved to be solved.

It is important to gain an understanding of the nature of the progression of a heating in order to improve the effectiveness of preventive and control measures, including inertisation and emergency evacuation. By understanding the factors in heating progression, it is possible to estimate where an open fire may be initiated and predict the time until this is imminent.

Results from bulk coal testing at The University of Queensland’s Spontaneous Combustion Testing Laboratory using a two-metre column (Figure 1) have demonstrated that heatings develop away from the edge and progress towards the oxygen rich side (Beamish et al., 2002). Monitoring of the off-gases produced by the coal heating provides a good indicator of when the development of an open fire is imminent.

The understanding of this migrating hot spot is relatively new and has potential to assist mine site and mines rescue personnel in better managing the risks of spontaneous combustion. An effective means to communicate the concepts and develop understanding is through visualisation of the hot spot development process.

A computer package was developed as a fourth year special topic in mining project to visualise temperature and gas data for The University of Queensland’s two-metre column. The aim of the project was to develop a 3D model of the equipment, and then to create an animation that visualised the changes in temperature and gas readings recorded from a column test over time. The package developed is called Hotspot and it helps the viewer to gain a better understanding of the way that a hot spot forms and migrates to become a major spontaneous combustion hazard. This paper presents the background to the operational inputs and outputs of the package as well as a series of snapshots from a test of a Bowen Basin bituminous coal.

THE HOTSPOT PACKAGE

The current version of Hotspot runs as a utility in the 3D Studio Max™ software environment. It uses the 3DSMAX architecture and the MAXScript programming language to create the visualisation animation. The utility loads the temperature and gas data from text files and performs a quick analysis. The results of the analysis may then be used to configure the visual environment by setting data ranges, selecting a time scale for the visualisation and setting other visualisation options. Once configuration is complete, the visualisation process may be executed. This produces an animated model in the 3D Studio MAX file format. This model can then be used to examine the data in real time, to render a distributable ‘.avi’ movie file or to make still pictures at any point in the visualisation.

Hotspot currently visualises temperature readings from the eight thermocouples at the centre of the column (Figure 1), the levels of carbon monoxide, oxygen, hydrogen, methane, carbon dioxide, ethylene and ethane in the off-gas, and also the Graham’s ratio of the off-gas stream. The visualisation allows for all this information to be seen together on one screen, changing over time. Used in this way, animation is a powerful and intuitive method of data visualisation for conveying data that spans time, indicates movement, or deals with multiple changing variables.

COAL SAMPLE AND UQ TWO-METRE COLUMN TEST PROCEDURE

Beamish et al (2002) describe the basic operation of the UQ two-metre column. The column has a 62 L capacity, which equates to 40 - 70 kg of coal depending upon the packing density...
used. The coal self-heating is monitored using eight evenly spaced thermocouples along the length of the column that are inserted into the centre of the coal at each location (Figure 1). Eight independent heaters correspond to each of these thermocouples and are set to switch off at 0.5°C below the coal temperature at each location so that heat losses are minimised and effectively semi-adiabatic conditions are maintained radially.

A sample of a Bowen Basin medium volatile bituminous coal was obtained from a larger batch of fresh run-of-mine (ROM) coal for testing in the UQ two-metre column. The coal particle size was kept below 150 mm, representing normal ROM coal and a size distribution of the sample was determined prior to loading into the column. The average particle size of the sample was 6.42 mm, based on the procedure described by Kunii and Levenspiel (1991) for estimating the surface-volume average particle size from the size distribution of the coal. Three subsamples were taken at this stage to obtain data on the as-received moisture of the coal.

The coal was loaded into the column with three 20 L plastic buckets. Once all the coal was in the column it was sealed and the heaters used to set the starting coal temperature of 25°C. This was achieved overnight. Air was then introduced to the coal at 0.5 L/min. A computer records all data at ten-minute increments. The column has several safety devices including computer-controlled trips on the external heaters and a temperature trip on the air inlet line. These were set to ensure maximum safety during operation of the column.

**SNAPSHOTS OF A HEATING**

A summary of the column test conditions is provided in Table 1. It must be remembered that the airflow used in this test is possibly an order of magnitude higher than that expected from air leakage, but it is still creates a low flow to mass ratio that would occur in a real heating.

### Table 1

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sample mass</td>
<td>62.6 kg</td>
</tr>
<tr>
<td>Packing density</td>
<td>996 kg/m³</td>
</tr>
<tr>
<td>Average particle size</td>
<td>6.42 mm</td>
</tr>
<tr>
<td>Initial moisture content</td>
<td>4.1%</td>
</tr>
<tr>
<td>Initial coal temperature</td>
<td>25°C</td>
</tr>
<tr>
<td>Airflow</td>
<td>0.5 L/min</td>
</tr>
<tr>
<td>Inlet air temperature</td>
<td>23 - 24°C</td>
</tr>
</tbody>
</table>

Snapshots from the Hotspot movie file of the test are presented for days six, 11, 13, 15, 16, 17, 18 and 19 in Figures 2 - 9 respectively. After the first six days the coal nearest the inlet has cooled to 23°C, while the coal approximately 1.5 m from the inlet has warmed up to 30°C (Figure 2). By day 11 the coal has warmed to almost 70°C at the same location and there is a noticeable increase in the Graham’s ratio, CO make and hydrogen evolution (Figure 3). A significant hot spot has formed by day 13, with the coal temperature reaching 130°C (Figure 4). By this stage ethylene is detected and the oxygen on the outlet begins to fall rapidly, due to the accelerated coal oxidation.

By day 15 the hot spot can be seen migrating towards the air source as the coal is now drier in this region and it is chasing the air to sustain the accelerated oxidation reaction (Figure 5). The gas levels are now increasing quite dramatically during this stage. By day 18 the hot spot has migrated to the bottom half of the column and also increased in size (Figure 8). By day 19 the hot spot has reached excessive temperatures (up to 270°C) and the test is terminated to prevent the coal reaching ignition.

![Fig 2 - Bowen Basin bituminous coal, day 6.](image)
COMPUTER ANIMATION OF HOT SPOT DEVELOPMENT IN BULK COAL AS AN AID FOR TRAINING COAL MINERS

FIG 3 - Bowen Basin bituminous coal, day 11.

FIG 4 - Bowen Basin bituminous coal, day 13.
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**FIG 5** - Bowen Basin bituminous coal, day 15.

**FIG 6** - Bowen Basin bituminous coal, day 16.
COMPUTER ANIMATION OF HOT SPOT DEVELOPMENT IN BULK COAL AS AN AID FOR TRAINING COAL MINERS

Fig 7 - Bowen Basin bituminous coal, day 17.

Fig 8 - Bowen Basin bituminous coal, day 18.
The results obtained from this UQ two-metre column test are completely consistent with the moist coal models published by Schmal, Duyzer and van Heuven (1985), Arisoy and Akgun (1994), Portola (1996), Monazam, Shadle and Shamsi (1998) and Akgun and Essenhigh (2001). The development stages of the hot spot, including the migration towards the air source, must be considered by mining personnel when assessing the risk of spontaneous combustion. By using the Hotspot package combined with individual testing of coal from each mine a better understanding of the spontaneous combustion risk can be obtained.

CONCLUSIONS

For the first time it is now possible to show mine personnel the stages that are associated with the development of a heating. Hot spot formation is not one continuous temperature increase, nor is it a stationary phenomenon. This feature of a heating is clearly shown in the results from bulk coal self-heating when they are loaded into the computer animation package Hotspot. This new development in spontaneous combustion visualisation provides a powerful training tool for mine personnel that can be tailored to show features relevant to the coal being mined. To achieve this, the coal must of course be tested in the UQ two-metre column using test conditions similar to that likely to be experienced at the mine. With the advances in computer visualisation technology it is envisaged that further enhancements of the Hotspot package will be possible in the near future that will provide even more benefit to the coal industry through competency training for recognition of relevant gas sampling frequency and analysis to track hot spot development. In this manner simulation exercises can be produced to assist with advanced learning.

ACKNOWLEDGEMENTS

The authors would like to thank the School of Engineering Development fund for sponsoring this project and ACARP project C12018 for supplying results to be used for the animation.

REFERENCES

Anatomy of a Heating and Assessment of Critical Self-Heating Parameters
D Humphreys

ABSTRACT
Numerical modelling is a valuable tool for simulating the fundamental processes that take place during a heating. The models presented in this paper have enabled a quantitative assessment of the effects of initial pile temperature, pile size and mass and coal particle size on the development of a heating. All of these parameters have a certain criticality in the coal self-heating process.

INTRODUCTION
Spontaneous combustion of coal has been a hazard to the mining industry from the very first attempts to mine coal.

The assessment of the propensity of coal for spontaneous combustion is largely based upon the results of any one of a large number of laboratory-based tests. Inevitably, there are compromises in the laboratory tests to increase the rate of oxygen self-oxidation. In situ. As a result, the assessment of the propensity of a coal for spontaneous combustion is reduced to a qualitative interpretation of the laboratory test results. Despite a plethora of different methods purporting to test the spontaneous combustion propensity of coal, too often the question arises ‘What does this mean?’ Quantitative assessment of the potential for spontaneous combustion in a coal pile is lacking. Numerical modelling offers distinct advantages to assess the anatomy of a heating and to quantify the effects of various intrinsic and extrinsic parameters.

BASIS OF NUMERICAL MODELLING
To better understand the complexities involved in the development of spontaneous combustion Humphreys (2004) developed a number of numerical models to simulate self-heating in one-, two- and quasi-three-dimensional models plus two commonly used laboratory tests. The multitude of factors that affect the oxidation of coal and therefore the development of spontaneous combustion have been extensively reviewed by many authors and will not be reviewed again here. However it is necessary to record that the numerical models took into account the impact of temperature, oxygen concentration, particle size, prior oxidation, the inherent reactivity, the heat of oxidation, and convective and conductive heat losses upon the gaseous interchanges taking and the temperature increases that occur during the development of spontaneous combustion in a coal pile.

To simulate self-heating, a volume of coal is represented by a series of interconnected nodes. Each node is taken to represent a discrete volume and mass of coal through which air passes and in which oxidation, and therefore heat generation, take place. The mass is assumed to be concentrated at the nodes and all reactions, oxidation, wetting and drying, are assumed to take place at nodes. All heat transfer (convection and conduction) processes occur between nodes.

A single line of nodes represents a one-dimensional model with airflow from node to node (plug flow). Heat transfer is also from node to node as illustrated in Figure 1, with no heat transfer perpendicular to the line of nodes. The only heat losses to the environment occur at either end of the model and therefore one-dimensional models are restricted in examining the effects of scale. This type of model simulates the behaviour of a column of coal in an infinitely wide slab of coal. As a one-dimensional model has the benefits of relative computational simplicity but is limited in its application to real scenarios.

A two-dimensional model can be made to provide for more complex heat transfer processes. Airflow is still assumed to be homogeneous plug flow from one end of the node grid to the other, but conductive heat transfer can take place across a line of nodes. Convective heat losses can also occur at boundary surfaces as well as the end surfaces. A two-dimensional model simulates a slice through a block of infinite width perpendicular to the plane of the nodal grid. Where an axis of symmetry exists across which there is no heat transfer (an adiabat), the nodal grid can be split to reduce the number of nodes in the simulation. The number of nodes required for a two-dimensional model is obviously far more than for a one-dimensional model, but the two-dimensional model is better suited to more complex geometries.

The complexity of modelling increases from one to two and then to three dimensions, but a method has been developed which allows a quasi-three-dimensional model to be developed from a two-dimensional model. This can be done by considering each node as representing a cylindrical shell, rather than a slice of constant thickness. In a homogeneous cylinder there is no heat flow tangentially, only axially and radially. By calculating the area used to determine conductive heat transfer between nodes based on this idea, the basic two-dimensional model can be made to simulate a cylinder of coal of definite dimensions and mass. The resulting model is referred to as the Quasi-3D model.

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FIG 1 - One-dimensional heat transfer model.
MODELLING RESULTS AND DISCUSSION

The relationship between $R_{70}$ and RIT test results and oxidation characteristics

Both the RIT and $R_{70}$ tests purport to indicate the propensity of coal to spontaneous combustion. These two tests were developed independently many years apart and are often used by Australian operators to assess the potential for spontaneous combustion of coal. Australian Coal Industry Research Laboratories (ACIRL) has tested many Australian coals using both methods (Symmington, 1999). The assessment of the propensity to spontaneous combustion is based upon the qualitative assessments that coals with $R_{70}$ greater than 1°C/h or RIT less than 70°C are highly prone or ‘troublesome’ coals. If the $R_{70}$ is less than 0.5°C/h, or the RIT is greater than 120°C the coals are regarded as having low propensity to spontaneous combustion or are regarded as ‘safe’.

The main properties of coals that vary from coal to coal and impact the greatest on likely self-heating characteristics of a coal are the inherent rate of oxidation ($K_o$) and the heat of oxidation ($H_o(40°C)$). The models developed to simulate the RIT and $R_{70}$ tests were used to examine the effects of these coal properties on the RIT and $R_{70}$ results. For a range of combinations of $K_o$ (from 2000 to 22 000 g O$_2$/min/kg coal) and $H_o$ (from 1000 to 15 000 J/g O$_2$), the $R_{70}$ and RIT models were run to determine the corresponding $R_{30}$ and RIT results. The relationships between these four parameters as indicated by modelling are shown in Figures 2 and 3.

From these results, it is clear that the two spontaneous combustion indices are affected in different ways. Consider the results obtained by modelling RIT as shown in Figure 2. The primary factor determining RIT is clearly the characteristic mineral-matter-free oxidation reactivity of the coal proportion, $K_o$. The heat of oxidation has only a minor effect on the RIT. Large variations in heat of oxidation ($H_o(40°C)$) from 1000 to 15 000 J/g oxygen adsorbed make very little difference to the RIT results obtained for a given rate of oxidation. On the other hand, it is clear from Figure 3 that $R_{70}$ is dependent on both coal properties.

The explanation for this can be obtained by considering the nature of the two tests in relation to the coal properties that are involved. Taraba et al (1988) showed that the low temperature heat of oxidation was a function of temperature. About above 80°C, the heat of oxidation for all coals was about the same (approximately 15 000 J/g O$_2$) but varied considerably from coal to coal at 40°C. Therefore in the RIT test, which begins at 70°C, there will be little variation in the heat of oxidation from coal to coal and it is reasonable to expect RIT to be independent of the low temperature heat of oxidation. This, however, is not the case in the $R_{70}$ test, which starts at 40°C. Therefore $R_{70}$ results will be influenced by the low temperature heat of oxidation and the oxidation reactivity. Clearly, this illustrates how important it is to understand any spontaneous combustion index in relation to the fundamental oxidation properties of coal to avoid the problems associated with purely qualitative assessments of spontaneous combustion.

Two additional models have also been developed to simulate the commonly used spontaneous combustion tests, the Adiabatic self-heating ($R_{30}$) test and the relative ignition temperature (RIT) test. These have been used extensively in Australia to characterise the spontaneous combustion of coals and a significant database of results exists. The numerical models for these tests simulate the conditions that apply to the tests by modelling the oxidation of coal in a single node with appropriate heat loss calculations for each test method (Humphreys, 2004).

It is reasonable to expect therefore that there is no direct correlation between RIT and $R_{30}$ and this in fact is the case when one considers the distribution of RIT and $R_{30}$ results obtained for a large number of Australian coals as shown in Figure 4. (Note however that the horizontal axis represents the self-heating period at 70°C from the $R_{30}$ test (SHP($R_{30}$)) or 30/R$_{30}$. This places the apparently more prone (higher $R_{30}$, lower RIT) in the lower left corner of the diagram.)

How then are the test results related to each other and to the principal oxidation parameters $K_o$ and $H_o(40°C)$? It is clear that there is not a simple correlation between RIT and $R_{30}$, but a more complex correlation involving the rate and heat of oxidation. The same data used to create Figures 2 and 3 has been plotted in Figure 4. This diagram shows lines of equal rate of oxidation and equal heat of oxidation. The actual test results fall within the range of reasonable values for the heat of oxidation, and there is an upper bound to the results when the low temperature heat of oxidation equals about 15 000 J/g of oxygen adsorbed. The results also provide a range for the rate of oxidation that will be useful in further modelling and a means of estimating the rate of oxidation and heat of oxidation based upon $R_{30}$ and RIT testing.

This analysis explains the apparent conundrum that a coal may rate as prone to spontaneous combustion when tested in the Adiabatic ($R_{30}$) apparatus (say $R_{30} = 1°C/h$, SHP($R_{30}$) = 30 hours) and less than ‘troublesome’ in the RIT apparatus (say RIT = 180°C).
Anatomy of a heating

The development of a spontaneous heating in coal is a complex phenomenon and is poorly understood, because it is so difficult to observe real heatings, particularly in underground coal mines, where they have the greatest potential to cause damage. Much of what is known about spontaneous heatings is derived from laboratory studies of very small-scale tests. These tests provide valuable data on the many parameters that affect the oxidation process and the production of off-gases that might be used for detection purposes. It is, however, virtually impossible to examine a real heating, to measure its temperature distribution, to measure the airflow involved, or to measure almost any aspect of its behaviour. It is impossible to conduct an ‘autopsy’ on a heating to see what has happened. Considerable insight can, however, be gained into the nature of real heatings by examining in detail the results obtained from a simulated heating.

Some understanding of the complexity involved can be obtained by examining the results of modelling the development of a heating in a pile of coal as simulated in the Quasi-3D model. The oxidation properties for the average test results shown in Figure 4 (R70 and RIT values of 0.90°C/hour and 162°C respectively) corresponding to values for Ko and Ho(40°C) oxidation reaction constants of 6500 g O2/min/kg and 4500 J/g O2 respectively. The relative oxidation rate was set to 25 per cent (equivalent to particle size of 91 to 400 mm) and the dimensions of the pile were set at 5 m diameter by 5 m long, simulating a thickness of the coal pile is insufficient, heat losses will and such a heating will not develop. For example, if the mass or may be circumstances in which a coal can be exposed to airflow dangerous heating to develop will be in reaching 125°C. There will be a reddish-brown oxide colouration on the surface of the coal, which is sufficiently large to pose a threat to safety will have to pass through the incipient phase. Most of the time required for a dangerous heating to develop will be in reaching 125°C. There may be circumstances in which a coal can be exposed to airflow and such a heating will not develop. For example, if the mass or thickness of the coal pile is insufficient, heat losses will

![FIG 4 - RIT versus SHP(R70) (ie 30/R70) versus rate of oxidation versus heat of oxidation showing tested results for Australian coals.](image)

1. The incipient phase characterised by peak pile temperatures up to about 125°C. During this phase a hotspot develops from the upwind surface, migrates downwind to a maximum depth and remains static in that position.
2. The migration phase characterised by the forward migration of the hotspot. During this phase the minimum oxygen concentration in the pile falls to zero per cent and there is a very rapid increase in the peak coal temperature. Without remedial action, the heating continues to develop and could lead to the outbreak of fire at the upwind surface of the pile.
3. The charring phase, when the temperature in the pile is sufficient to cause the formation of unreactive char or the outbreak of open fire.

The most significant phase in any heating is the initial incipient phase up to about 125°C. Any spontaneous heating which is sufficiently large to pose a threat to safety will have to pass through the incipient phase. Most of the time required for a dangerous heating to develop will be in reaching 125°C. There may be circumstances in which a coal can be exposed to airflow and such a heating will not develop. For example, if the mass or thickness of the coal pile is insufficient, heat losses will

The main features of the development of the heating in the base case are illustrated Figure 5. The peak temperature in the pile, the position of the peak temperature (the hotspot) and the position of the minimum oxygen concentration in the pile are shown in Figure 5. At the very start of the heating, there is a moderately rapid increase in temperature, with the ‘hotspot’ located at the upwind surface of the pile and the minimum oxygen concentration at the back surface. The rate of temperature rise moderates (not visible on the figure but occurs nevertheless) and the position of the peak temperature moves gradually downwind. The position of the minimum oxygen remains at the back surface of the pile, although the minimum oxygen concentration is decreasing. After 275 hours, the peak temperature has moved to the furthest downwind position at about 1900 mm. The hotspot remains in this position until its temperature exceeds about 125°C.

This triggers a change in the behaviour of the heating and the hotspot begins to migrate forward. Shortly afterwards, the minimum oxygen concentration in the pile falls to zero as does the oxygen concentration at the hotspot. Despite this, the peak pile temperature is increasing rapidly, at approximately 8°C/hour.

Once the positions of the peak temperature and minimum oxygen concentration coincide, they begin to migrate together toward the upwind surface. This can only begin when the temperature profile in the coal ahead of the hotspot is sufficient to consume all the oxygen entering that part of the pile. The forward migration of the heating is limited by the upwind surface, which triggers another increase in the coal temperature. A short while after this, the temperature of the coal is sufficient to cause charring and a charline is formed in the pile. The final phase of the heating is the lateral expansion and downwind migration of the charline, as all the reactive elements in the coal are consumed by oxidation.

From this analysis, it is possible to divide the development of this heating into three distinct phases:

1. The incipient phase characterised by peak pile temperatures up to about 125°C. During this phase a hotspot develops from the upwind surface, migrates downwind to a maximum depth and remains static in that position.
2. The migration phase characterised by the forward migration of the hotspot. During this phase the minimum oxygen concentration in the pile falls to zero per cent and there is a very rapid increase in the peak coal temperature. Without remedial action, the heating continues to develop and could lead to the outbreak of fire at the upwind surface of the pile.
3. The charring phase, when the temperature in the pile is sufficient to cause the formation of unreactive char or the outbreak of open fire.
predominate at some temperature and a spontaneous heating will not occur. However, for heatings of significance, the incipient phase will be significant and there are a number of important factors in determining whether spontaneous combustion will occur in a particular coal.

Factors controlling the occurrence of spontaneous combustion

One of the main objectives of this study was to examine the nature of spontaneous heatings in coal and to relate the observed spontaneous combustion behaviour to measurable coal properties. Much of what is already known is based on an incomplete understanding of the phenomenon of spontaneous combustion.

What is poorly understood is the degree to which changes in many parameters affect the spontaneous combustion behaviour of coal. For example, consider the thickness of a coal seam. If it is assumed that the coal is broken up to a certain size and that there is airflow through the coal, how will the thickness of the seam affect spontaneous combustion behaviour? It could easily be accepted that if the seam is only 100 mm thick, there will be no spontaneous combustion, due to heat losses to the surrounding unreactive strata. But what will occur if the seam is 1 m thick, or 2 m or 3 m? As the coal seam thickness increases it could be expected that the heat losses relative to the heat generated by oxidation decrease and lead to higher and higher temperatures. At some critical thickness the temperature will be such that a heating will develop. There need be no changes in any of the other parameters for this to occur.

The same can be said of all the other parameters that have been listed. Any change which leads to greater heat generation, or lower heat losses, will cause the temperature in the coal pile to increase more rapidly and, at some critical value, will cause spontaneous combustion to occur.

The critical airflow flux to trigger spontaneous combustion

Airflow is essential for spontaneous combustion to take place. In any coal pile, without airflow through the pile, there is no oxygen available for oxidation and there can be no temperature rise. It has often also been said that high airflow is capable of carrying the heat generated by oxidation away from the pile and therefore can be used as a strategy to help control spontaneous combustion.

To examine the impact of airflow on the development of a spontaneous heating in a coal pile, a series of models using a one-dimensional model was run to simulate heating in a coal pile, such as a coal pillar in an underground mine. The airflow flux was altered over a wide range from 0.1 L/min/m² to 5000 L/min/m² with other model parameters the same as that used in the previous base case simulations.

It was found that at very low airflow flux levels, there is a temperature rise but the development of the heating restricted by the lack of oxygen entering the pile. Below about 3 L/min/m², the airflow flux has a significant effect on the development of spontaneous heating. Above 3 L/min/m², this effect becomes less significant and there is very little impact upon the development of the heating. For the purposes of additional modelling, it was decided therefore to use an airflow flux of 20 L/min/m² to ensure there was no effect due to inadequate flow of air through the model.
It was also found that the impact of increasing airflow through the pile was to force the position of the hotspot formation further downwind in the pile. However, only if extreme values were simulated did the heating move significantly into the pile suggesting that it was not possible to rely on excess ventilation to control self-heating as has been suggested previously.

**Critical spontaneous combustion characteristics**

As discussed above one could expect that there is a critical thickness associated with a particular coal and size below which it would not be possible for spontaneous combustion to occur. The relatively high heat losses when the seam is thin prevent the necessary temperature rise required for transition from the incipient phase to the migration phase.

The impact of increasing seam or pile thickness was modelled using a variation on the 2D model to simulate the self-heating of coal in a deep pile of coal of varying thickness with the same coal properties as the base case. The initial thickness in the model was incremented until there was little change in the time-temperature profile. The results obtained are illustrated in Figure 6. As expected, for thin layers of coal, there is an initial temperature increase to some maximum value after which the coal begins to cool due to reduced rates of oxidation and heat transfer into the surrounds. As the coal thickness increases, the maximum temperature achieved before the coal begins to cool increases and the time at which this maximum is achieved increases. Up to some thickness, in this case about 1.5 m, the increase in oxidation rate due to increased temperature has been more than offset by the decrease caused by cumulative oxidation. Once the coal temperature exceeds about 70°C, however, a fine balance is achieved between these competing effects. Any further increase in temperature will see the domination of the effect from temperature. The result is that, above some critical thickness, the effect of temperature eventually dominates and there will be a very rapid temperature rise beyond 70°C. The critical self-heating thickness for the conditions modelled and illustrated in Figure 6 is 1.5 m. For the purposes of this study, the critical self-heating thickness is the maximum thickness for a given combination of coal properties (particle size, heat of oxidation, oxidation rate constant and initial coal temperature) at which self-heating to 125°C will not occur.

**Critical self-heating period for relative oxidation rate of 25 per cent (particle size of 91 to 407 mm) and initial temperature of 35°C.**

![Critical self-heating period for relative oxidation rate of 25 per cent (particle size of 91 to 407 mm) and initial temperature of 35°C.](image)
If the coal thickness is greater than the critical self-heating thickness, spontaneous combustion will take place and very high temperatures will result. As the pile or seam thickness increases, the time required for the onset of rapid self-heating becomes less and less. As illustrated in Figure 6, this approaches some minimum value, shown as the critical self-heating period. In this case, the critical self-heating period is 550 hours. For the purposes of this study, the critical self-heating period is the minimum time taken for a given combination of coal properties (particle size, heat of oxidation, oxidation rate constant and initial coal temperature) to self-heat to 125°C.

Similarly, variations in the total coal mass can determine whether a heating will develop and affect the time required for the heating to develop. Clearly, the same logic applies when considering the impact of increasing the mass of coal in a pile. For very small piles, there will only be a small temperature rise, until heat losses balance heat generation. As oxidation proceeds, the rate of oxidation will decrease and the temperature of the small coal pile will begin to decrease. As the pile size increases, the maximum temperature achieved will increase until, eventually, the effect of increased oxidation is dominated by that of increased temperature and a fully developed heating with high temperatures will occur. For every combination of coal properties (particle size, heat of oxidation, oxidation rate constant and initial temperature) there will be a critical self-heating mass, which is the minimum mass of coal required to support self-heating to 125°C.

The critical spontaneous combustion characteristics of interest can therefore be defined as:
- the critical self-heating period (CSHP) – the minimum time in which spontaneous combustion will occur;
- the critical self-heating thickness (CSHThk) – the thickness of a coal pile or seam below which it is not possible for spontaneous combustion to occur; and
- the critical self-heating mass (CSHM) – the mass of a coal below which it is not possible for spontaneous combustion to occur.

The variation of critical self-heating period and thickness with oxidation properties can be assessed from Figures 7 and 8. In each of these figures the upper graph reflects the variation due to oxidation properties of the coal in the modelled critical self-heating period or thickness for coal with a relative oxidation

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**Figure 8 - Critical self-heating thickness for relative oxidation rate of 25 per cent (particle size of 91 to 407 mm) and initial temperature of 35°C.**
reactivity of 25 per cent (95 per cent passing 91 to 407 mm) and an initial temperature of 35°C. The position of the average coal \((K_0 = 6500, \text{Ho}(40°C) = 4500)\) is also shown for reference. The lower graph shows the same critical self-heating parameter as a function of \(R_{70}\) and RIT.

It is clear from these results that the both critical self-heating characteristics decrease with increasing oxidation reactivity \((K_0)\) and heat of oxidation at 40°C \((\text{Ho}(40°C))\) in a manner very similar to that of the critical self-heating period. It is also clear that neither \(R_{70}\) nor RIT results can claim to be a predictor of the critical self-heating period or thickness but that in combination these parameters can be used.

It should also be noted that it is predicted that some coals will not self-heat under the conditions modelled. The critical self-heating period and thickness are very high (infinite) and this is the case not matter how the particle size (relative reactivity) varies for these coals. This comes about because of the temperature dependence of the heat of oxidation suggested by the work Taraba (1988). Extrapolating his results to lower temperatures suggests that the heat of oxidation may become very low or zero. Therefore below some temperature, the critical self-heating temperature, it is likely that a particular coal is incapable of self-heating irrespective of the airflow, pile thickness, particle size, etc.

This behaviour is analogous to that seen in the USBM minimum self-heating temperature test described by Smith and Lazarr (1987) except that the critical self-heating temperatures are much lower than suggested by that test method. The relationship between the estimated critical self-heating temperature and the results of spontaneous combustion testing is shown in Figure 9. This adds one more parameter to the critical self-heating characteristics of coal being:

- the critical self-heating temperature (CSHT) – the initial coal temperature below which coal will not spontaneously combust irrespective of the thickness or mass of coal involved.

### Use of the critical self-heating characteristics to assess the potential for spontaneous combustion

It is considered that the critical self-heating characteristics of temperature, thickness and period, as described above, can be used to provide a better means of assessing the spontaneous combustion potential in a mining, or stockpiling situation. It is quite apparent, based upon the modelling results obtained, that if the initial temperature of a coal seam in an underground mine is lower than the critical self-heating temperature of the seam, spontaneous combustion will not occur. Similarly, if the seam
thickness is less than the critical self-heating thickness self-heating will not occur. This logic can be expressed in a simple flow chart as shown in Figure 10. This method of assessment represents a considerable improvement over the qualitative assessments of most spontaneous combustion test methods which leave the mine owner pondering the difference between high and low propensity to spontaneous combustion.

Obviously, caution must be used if it is predicted that there is no potential for spontaneous combustion and there is only a small difference between in situ properties and the critical self-heating characteristics of a coal. Small changes to in situ conditions could result in a different evaluation of the potential for spontaneous combustion. Caution must also be applied when the initial coal temperature is close to the critical self-heating temperature. Small variations in any of the coal properties especially determination of R70 or RIT and determination of the in situ initial coal temperature could have a significant impact on the outcome of the assessment of the spontaneous combustion potential. The method of assessment proposed here, however, is a considerable improvement over that which could be achieved if the only considerations were the results of the R70 and RIT results or any other qualitative interpretation of some laboratory test method.

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**Evaluation of Spontaneous Combustion Potential**

1. **Determine R and RIT, Initial Coal Temperature, Ash, Particle Size Distribution and Thickness of Seam or Pile**
2. **Determine Ko and Ho from R70 and RIT**
3. **Determine Critical Self-Heating Temperature from Ho**
4. **Does Initial Coal Temperature Exceed Critical Self-Heating Temperature for This Coal?**
   - No – Spontaneous combustion will not occur
5. **Yes – Determine Critical Self-Heating Thickness from Ko, Ho and Size Distribution**
6. **Does Seam/Pile Thickness Exceed Critical Self-Heating Thickness?**
   - No – Spontaneous combustion will not occur
   - Yes – Determine Critical Self-Heating Period
7. **Is the Coal Exposed to Air for a Period Exceeding the Critical Self-Heating Period?**
   - No – Spontaneous combustion will not occur
   - Yes – Spontaneous Combustion Potential

**Fig 10 - A method of assessing the potential for spontaneous combustion in a mine or stockpile.**
CONCLUSIONS
This study has attempted to combine a comprehensive knowledge of the many aspects of oxidation behaviour of coal and the heat loss mechanisms that play their part in spontaneous combustion to better assess the likely in situ or field behaviour of coals leading to the development of spontaneous heatings. This knowledge has been combined into a number of numerical modelling techniques that can simulate laboratory tests such as the R70 and RIT tests and to model the spontaneous combustion behaviour of coal in stockpiles, or in a mine. Fundamental oxidation parameters of reactivity and heat of oxidation are used as the basis for all models and therefore a direct link is established between results of the laboratory tests and predicted field behaviour.

The study has provided a fresh understanding of many aspects of the development of spontaneous combustion in coal leading in turn to the development a new more quantitative assessment of the potential for spontaneous combustion that better takes into account factors such as the initial pile temperature, pile size and mass, and coal particle size.

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The views expressed in this paper are those of the author.

REFERENCES
DEVELOPMENT OF LONGWALL GOAF GAS FLOW MODELS

Gas flow migration in a longwall goaf is complicated process as many factors are involved, such as ventilation layout and intensity, gas emission rate and compositions (e.g., the presence of methane and carbon dioxide), face (seam) orientation and dip, gas buoyancy and goaf permeability. A range of CFD models have been developed to achieve a detailed understanding of the gas flow mechanics and distribution in longwall goafs. In addition to innovative CFD modelling, the study also involved gas management strategies but also the control of spontaneous combustion risk in the goaf. Work is continuing to develop general guidelines of proactive goaf inertisation strategies to suppress the development of spontaneous heating behind longwall faces.
extensive validation and calibration of initial models using data obtained from field studies and parametric studies to investigate the effect of various parameters on goaf flow patterns. Models were then used in the development of gas and spontaneous heating control strategies through simulation of the effectiveness of various designs and control techniques. The CFD modelling work generally consists of a number of key stages, including:

- field studies to obtain the basic information on panel goaf geometries and other parameters;
- construction of 3D finite element model of the longwall goaf;
- setting up flow models and boundary conditions through UDFs;
- base case model simulations;
- model calibration and validation using field measured data; and
- extensive parametric studies and development of optimum strategies.

A key part of the CFD models is the incorporation of longwall goaf permeability distributions and gas emissions via a set of UDFs that are linked to the FLUENT solver. Flow through the goaf was handled using custom written subroutines, which were added to the ‘flow through porous media’ modules of the basic code. In these subroutines/modules, flow through the porous goaf regions was simulated by adding a momentum sink to the momentum equations. The sink had a viscous part proportional to the viscosity and an inertial component proportional to the kinetic energy of the gases. A number of subroutines were written to represent different ventilation and goaf gas emissions (gas drainage) scenarios, which were then combined with the main FLUENT program to carry out the simulations.

The distribution of goaf permeability was derived from results of extensive previous studies and longwall geomechanics models. Pressure, flow rate and gas distribution in a typical longwall goaf were used to calibrate the initial models and further refine the distribution of goaf permeability. Pressure, flow rate, gas concentration measurements and tracer gas study results in a typical longwall panel were used to further refine the distribution of permeability. A standard two-equation k-e model was used to estimate the turbulent transport through the flow region and the flow near the boundaries was approximated by the use of standard wall functions. The models were set up to simulate both turbulent flow conditions near the face and laminar flow inside the goaf region.

CFD longwall models can be developed according to the actual mine layouts, as shown in Figure 1, and hexahedral cells are commonly used as this enables greater accuracy of boundary layer calculations and the ability to stretch the blocks along roadways. The mesh used in the models was ‘refined’ with higher density mesh in the areas of interest such as areas next to the face and roadways. A typical geometry and mesh used in longwall goaf gas flow models is shown in Figure 2.

LONGWALL PANEL VENTILATION STUDIES

The effects of poor mine ventilation are far-reaching and can result in problems with high gas emissions and spontaneous combustion. An analysis of face/panel ventilation systems would be useful to assess the potential of spontaneous heating and any changes in mining conditions or mine design which may lead to such hazards.
A major feature of CFD modelling is its capability to predict what will happen under a given set of circumstances, i.e. it can answer many "what if?" questions before a proposed design is implemented in the field. CFD models have been developed and used as investigative tools to provide preliminary prediction on goaf gas flow patterns based upon proposed ventilation arrangements, as well as an opinion on spontaneous combustion risk and optimum goaf gas drainage to help gas control in the face.

Figure 3 shows the application of CFD simulation in this area. The figure shows the oxygen distribution patterns under different panel layouts, face orientation and gas emission conditions. Information from these studies is useful in understanding the magnitude of oxygen ingress into the goaf areas and hence the potential for spontaneous heating under different ventilation designs. Such an understanding would be helpful in the selection of an optimum face ventilation designs that would not only allow the control of goaf gas emission but also the minimisation of spontaneous heating occurrence.

GOAF GAS CONTROL AND SPONCOM PREVENTION

A significant contribution from the CFD modelling work has been the development of innovative goaf gas control strategies in a highly gassy, Australian underground colliery with a propensity for spontaneous combustion.

The major factors that influenced spontaneous heating and CO production rates at the mine site included: ventilation design, seam structure/gradients, caving pattern behind the face, length of back return, location and orientation of faults/dykes, and condition of gateroads immediately behind the face. It was observed that although the effect of some of these geological factors on face control was minimal, they had a major influence on goaf gas flow dynamics and the occurrence of goaf heating.

CFD modelling was carried out to study the effect of the following parameters:

- open intake gateroad in the goaf on oxygen ingress,
- increased permeability due to dykes on gas flow dynamics,
- U ventilation system on gas distribution,
- 100 per cent CH₄ gas on goaf flow dynamics/buoyancy, and
- other goaf gas drainage optimisation studies.

An example of the effect of a partially open intake gateroad (simulating the effect of strong supports such as ‘CANs’) is shown in Figure 4. Results showed that an open intake gateroad in the goaf increases the oxygen penetration up to 300 to 400 m behind the face.

A particular application of CFD modelling has been the optimisation of surface goaf gas drainage designs with the objective to maximise the capture of goaf gas whilst minimising the risk of spontaneous combustion. Figure 5 shows the predicted oxygen ingress patterns into the goaf with different goaf gas drainage layouts.

Some of the innovative control strategies resulting from goaf gas control studies, which have a major impact on reducing spontaneous heating risk include:

- new goaf hole designs to ensure that oxygen concentration in the holes was below four to five per cent;
- goaf holes at 80 to 100 m away from fault/dyke areas;
- immediate sealing-off of the cut-throughs behind the face (only one cut-through open for back-return);
- reduction in air velocity on the intake side of the goaf;
- uniform and continuous operation of goaf holes (sudden peaks and lows in goaf drainage flow rate increases the sponcom risk).

These strategies, together with a set of guidelines for optimum goaf drainage strategies has been successfully implemented at several Australian coal mines, including Dartbook, Central and Appin, which will prove invaluable in helping other mines improve their strategies.
In underground gassy coal mines it is generally recognised that immediately after sealing a longwall panel, the atmosphere behind the seals may enter and pass through the explosive range. The duration of explosive conditions in the sealed longwall goaf ranges from a few hours to one or two days or even a few weeks, depending on the gas emission rate and goaf characteristics. Therefore, any sealed area with methane as the seam gas has the potential to explode depending on the presence of ignition sources. To minimise this risk of explosions, the modern practice in some of the Australian mines is to inject inert gas into the sealed goafs immediately after sealing the panel. The specific objective of inert gas injection operations is to reduce the goaf oxygen levels below the safe limit of eight per cent (ie with a safety factor of 1.5 over the explosive nose limit of 12 per cent) before methane concentration reaches the lower explosive limit of five per cent.

Traditionally inertisation schemes usually involved just injecting inert gas through maingate (MG) or tailgate (TG) seals until goaf gas sampling results show that the oxygen level was below eight per cent. In many cases it was found that the goaf oxygen concentration was above 12 per cent even after two to three days of inert gas injection and in some cases an explosive atmosphere was also present in the goaf during inertisation.

CFD models have been used to develop optimum and effective strategies for inertisation during longwall sealing operations to achieve goaf inertisation within a few hours of sealing the panel. Again this study has combined the detailed analysis of the performance of various inertisation field trials together with CFD modelling results of different inertisation operations in order to develop the optimum inertisation strategies.

A number of parametric studies were conducted on the base case CFD models that had been calibrated and validated based on the information obtained from previous inertisation studies and goaf gas monitoring. These studies included changes in inert gas injection locations, inert gas flow rates, seam gradients, and different inertisation strategies to investigate their effect on goaf inertisation. Parametric studies were conducted under both steady state and transient conditions.
The modelling results show that there were no major differences in goaf gas distribution between the injection of boiler gas and nitrogen; however, different inert gas injection points resulted in entirely different goaf gas distribution. Figure 6 shows oxygen distribution in the goaf for inert gas injection at different locations after 24 hours of injection. Inert gas at a rate of 0.5 m³/s was injected through the MG seal and at 200 m behind the face (through 3 c/t seal) on the maingate side respectively.

Analyses of the figures indicate that the strategy of inert gas injection through the MG seal was not as effective as the alternative strategy of inert gas injection at 200 m behind the face (ie through 3 c/t). Analysis of the various simulation results also indicated that longwall panel geometry, goaf characteristics, gateroad conditions in the goaf, goaf gas emission rates and composition, ventilation during panel sealing off period, chock withdrawal and panel sealing sequence would also have a significant influence on goaf gas distribution and inertisation.

The optimum inertisation strategies have been implemented at Newlands Colliery and were highly successful in converting the goaf environment into an inert atmosphere within a few hours of panel sealing.

PROACTIVE INERTISATION

‘Prevention is better than cure’. On the basis of previous studies, an on-going project at CSIRO is the development and demonstration of proactive inertisation strategies with the objective to reduce the risk of spontaneous heatings in active longwall faces, in particular under unexpected scenarios such as during slow retreat/face stoppage due to difficult geological conditions. Figure 7 shows the steady state results of a preliminary CFD study of inert gas (boiler gas) injection at 10 m and 60 m on the maingate side respectively behind the face during ‘normal’ face retreat. The results indicate that inert gas injection at 60 m behind the face is more effective in narrowing the high oxygen zone thus reduce the risk of spontaneous heating. Further studies in this area are continuing.

CONCLUSIONS

In combination with field studies, CSIRO have conducted extensive CFD modelling work to investigate the gas flow mechanisms within longwall goafs. These studies have greatly improved the fundamental understanding of goaf gas flow patterns and gas distribution in the longwall goaf and thus help...
the development of innovative gas control, spontaneous heating prevention and goaf inertisation strategies. Further investigations are continuing in a number of areas, including the study of proactive inertisation strategies to reduce the risk of heatings in the active longwall goafs.

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Detection of Underground Spontaneous Combustion of Coal With Surface-Based Radon Technique at Dartbrook Mine

S Xue¹ and N Winkelmann²

ABSTRACT

Underground coal heating is a significant hazard in some coal mines. The broad principles of how underground coal heatings occur are reasonably known, however the technique for locating the heatings in often inaccessible goaf has been proved to be difficult. Since 2002, CSIRO, with the support of the Australian Coal Association Research Program (ACARP) has been undertaking investigations to apply and develop innovative surface-based radon technique for locating the underground heatings in Australia. This paper presents the progress of the investigation.

RADON TECHNIQUE

The surface-based radon technique was initially developed by Taiyuan University of Technology, China in 1992. The technique was then commercially used to locate underground heatings in 1995. Since then, it has been used to locate more than 30 underground heatings in China, and great success has been claimed.

Principles

Radon-222 (Rn-222) gas occurs naturally as a decay product of the long-lived uranium-238 that is a common rare element in rock/coal/soil strata. Rn-222’s emanation ratio from strata is influenced by many factors such as its lithology, mineral particle size, porosity, surrounding stress, ground hydrology, and temperature. However the degree of the influence by these factors is quite different on different types of rocks/coal/soils.

Limited experimental investigations indicate that the temperature has a significant influence on Rn-222’s emanation ratio from coal. Figure 1 shows the experimental results of temperature dependence on the emanation ratio from coal for the range of 20°C to 260°C.

The transport of radon and other gases through the earth is a well documented phenomenon. For example Kristiansson and Malmqvist (1982), demonstrated that a radon anomaly could be located at the surface above a strong radium source placed in a mine some 150 m vertically below with intermediate strata of quartzite and shale. Since then many studies have demonstrated that the earth is continuously exhaling gases, including short-lived radon (t1/2 = 3.8 days). Mineral exploration makes use of the gas transport by looking for weakly adsorbed trace metals in soils which are carried by the gas to the earth’s surface.

More contentious has been the determination of the mechanism that permits and drives this gas transport. The most widely recognised theory considers that the flow takes places as microbubbles driven by gas pressure differences. These gas bubbles, comprising mainly CO₂ and CH₄ could then carry other rare gases (eg radon) as well as Au, Cu, Pb, Zn, etc (Zhou et al, 2003). Several laboratory tests have been reported to confirm that small gas bubbles can in fact act in the proposed manner (Varhegyi et al, 1992; Etiope and Lombardi, 1996).

Many authors still report the transport of gas over large distances through seemingly impervious rock layers as ‘strange’. Radon, in particular is the most puzzling as its transport must happen very rapidly and the driving force for such rapid flow is difficult to imagine. An alternative model has been recently proposed to explain rapid movement which is based on quantum mechanics and models the behaviour of a particle bound in a potential well (Holub and Smrz, 2002). Deformation of the well

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within impenetrable walls of a microcrack leads to the probability of localisation of the particle near the end of the crack. These authors have also identified the common occurrence of geoaerosols (Holub et al., 2001) in gases emanating into caves. These aerosols carry both radioactive and inert elements into the air space. One aspect of these geogases is that radioactive decay and radon play a fundamental role in their formation and release. Suggestions have been made that it is radioactive decay that causes the nucleation centres in formation waters, supersaturated with gases.

**Operation**

The technique employs an alpha detector and alpha cups. The detector is a portable, battery powered ‘alpha counting’ detector with an ionisation chamber for detecting alpha radiation of radon and its daughters. The cup, also called ‘alpha cup’, is an open-end plastic cup with a sorbent coating on its internal surface. Radon and its daughters are adsorbed and deposited on the internal surface of the cup. The working principle of the detector is shown in Figure 2, and the alpha cup and radon detector are shown in Figures 3 and 4 respectively.

The operational aspects of the technique are fairly simple. The operation steps include:

1. select the detection area and design surface grids of measurement points;
2. dig holes at the points (about 30 cm in diameter and 40 cm in depth) and place alpha cups upside down;
3. bury the cups for at least four hours and then recover them for on-site measurement with the detector; and
4. process the measurement data with a specially developed software.

**DEMONSTRATION OF THE RADON TECHNIQUE IN AUSTRALIA**

In 2002, the Australian Coal Association Research Program (ACARP) sponsored a project C12005 to investigate and demonstrate the radon technique. The main objectives of the project were to:

1. investigate the technique and gain a detailed understanding of the technique and its applications; and
2. demonstrate the technique at Dartbrook mine.

The project was successfully completed in September 2003 and the details and results of work carried out under the project were reported to ACARP (Xue, Balusu and Worrall, 2003). The main conclusions are summarised below:

1. The specific principle of the radon technique for locating underground heatings from surface needs to be further investigated.
2. The radon technique was demonstrated in two areas of suspected heatings at Dartbrook mine. The first one covers the surface area of 103 395 m² above LW7 goaf. The cover depth in the area was about 340 m. The second area of 10 000 m² was centred at CDH003 hole above LW2-3 goaf. The cover depth in the area was about 200 m. Five heating zones were identified with radon technique in the areas. Figure 5 shows the surface area above LW7 and Figure 6 shows the corresponding underground heating zones identified with the radon technique within the area. The results were consistent with those based on analyses of gas monitoring data and mining sequences.
3. It was also identified from the study that to apply the radon technique in the Australian coal mines with a greater
confidence, the detailed studies on the temperature dependence of radon emanation ratio from coal and its upright movement in overlying strata should be undertaken in the future research projects.

FUTURE APPLICATION OF THE RADON TECHNIQUE IN AUSTRALIA

In recognising the importance of the recommended studies, ACARP is funding a current radon project C13021. The project aims to:

1. Investigate temperature dependence of radon emanation from coal and effect of heatings on radon vertical movement in overburden strata. The dependence and effect are the core science upon which the radon technique for locating coal heating is based. Bench-scale testing will be carried out for the study.

2. Develop a portable prototype of radon detector and collector suitable for field measurement of radon measurement in mines.

3. Evaluate the prototype by re-surveying the surface radon flux over the goaf at Dartbrook mine and one other heating event if the opportunity arises.

The project commenced in April 2004 and is progressing well. A bench-scale test rig is being set up. The rig includes test apparatus, controlling system (temperature and air flow) and data sampling system. The portable prototype of radon collector and detector has been designed and ordered for manufacturing. The literature review of the radon movement theories is also underway. The project is scheduled to be completed in December 2005 and the details and results of work to be carried out under the project will be reported to ACARP by then.

REFERENCES


Economic Returns From Environmental Problems — Breeding Salt and Stress Tolerant Eucalypts for Carbon Sequestration, Salinity Abatement and Commercial Forestry

G Dale

ABSTRACT

Carbon sequestration in planted forests provides an immediately available, low cost option to address the greenhouse impacts of coal mining and coal utilisation in a carbon constrained world. In addition, planted forests also offer the opportunity to address other environmental issues, particularly salinity and loss of biodiversity.

Given salinity is characteristically associated with agriculture in the 500 to 700 mm rainfall belt, then where such forests can also produce commercial timber products, they offer the additional opportunity to establish new, diversified rural employment in both timber growing and processing.

In the majority of Australian catchments affected by salinity, where rainfall is generally less than the 700 mm limit to conventional forestry, and where groundwater reserves are often saline, the objective of achieving economically viable forestry production presents a significant challenge.

In 1996, Saltgrow commenced a collaborative breeding program to combine the salt and stress tolerance of *E. camaldulensis*, with the growth rate, stem form and wood properties of the commercial species, *E. grandis* and *E. globulus*. This program aimed to produce trees with the potential for commercial rehabilitation of saline landscapes. Results covering a range of site conditions from Saltgrow’s network of over 100 trials around Australia are presented. The opportunity for commercial plantations integrated with agriculture in low to medium rainfall areas to address salinity and sustainability will be discussed. In addition, the significant potential for partnerships between land-rich, cash-poor farmers, catchment management authorities seeking to invest in public good projects that enhance environmental sustainability, and major carbon producing industries with either mandated or voluntary requirements to curtail net emissions will also be discussed.

INTRODUCTION

In the 2001 - 2002 period, Australia exported just over 200 million tonnes of black coal†. Based on an average carbon content for black coal of 75 per cent, this represents 150 million tonnes of carbon, or 550 million tonnes of potential carbon dioxide emissions after combustion. Similarly, domestic consumption of black coal in the 2001 - 2002 period was around 72 million tonnes‡, representing potential CO₂ emissions after combustion. Similarly, domestic consumption for black coal of 75 per cent, this represents 150 million tonnes of black coal †. Based on an average carbon content for black coal of 75 per cent, this represents 150 million tonnes of black coal. Given salinity is characteristically associated with agriculture in the 500 to 700 mm rainfall belt, then where such forests can also produce commercial timber products, they offer the additional opportunity to establish new, diversified rural employment in both timber growing and processing.

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At the opening of BHP’s Dendrobium colliery at Mt Kembla in November 2003, NSW Premier Bob Carr stated that two Australian Coal suppliers had been told by European buyers that they wished to purchase coal with greenhouse offsets§. With ratification of the Kyoto Protocol by Russia on 18 November 2004, and entry into force of the protocol from 16 February 2005, the pressure from overseas coal purchasers for supply of coal together with greenhouse offsets can be expected to increase. All other factors being equal, suppliers that can provide greenhouse offsets at the most competitive price can be expected to enjoy a market advantage. Similarly in Australia, the NSW Benchmarks scheme has placed pressure on NSW power retailers to reduce CO₂ emissions. It is likely that other states will follow this lead.

Numerous technologies exist to improve coal generation efficiency, to capture and store CO₂ following combustion, to achieve demand side efficiencies, and to avoid net carbon emissions via renewable energy generation. Detailed discussion of these alternatives is beyond the scope of this paper other than to say that many of these options remain developmental or cost prohibitive. In contrast, carbon sequestration via planted forests offers an immediately available, cost effective means of providing carbon offsets, either against domestic consumption, or attached to coal exports.

In addition to their carbon sequestration potential, planted forests offer the potential to address many other environmental issues. Salinity currently affects over 5.6 million ha across Australia, and this is projected to increase to 17 million ha by 2050 without intervention (NLWRA, 2000). Of the projected area to be affected, over 13.6 million ha or 80 per cent is agricultural land (NLWRA, 2000).

Salinity is generally regarded as resulting from clearing of native deep rooted perennial vegetation and its replacement with annual crop and pasture species. This vegetation change, and the associated differences in plant water-use, has lead to an altered water-balance and rise in groundwater tables, ultimately bringing salt stored deep in the soil profile to the surface. Commercially driven tree production systems developed for large areas of the current crop and pasture zones of the Murray Darling Basin is one of three pillars of on-ground action recommended to halt the growth of salinity and loss of native biodiversity in Australia’s land and river systems (Stirzaker et al, 2000).

However, in the majority of catchments affected by salinity, where rainfall is generally less than the 700 mm limit to conventional forestry, and where groundwater reserves are often saline, the objective of achieving economically viable forestry production presents a significant challenge.

The XYLONOVA Research and Development Program commenced in 1996 with the aim of developing salt and drought tolerant eucalypt hybrids for establishing commercial plantations under low rainfall conditions and on saline, and waterlogged land. The program’s primary objective was to combine the salt and drought tolerance plus timber characteristics of *Eucalyptus camaldulensis* with the growth rate, wood quality and form of *E. grandis* and *E. globulus*. To date 1333 novel varieties have been developed, and over 100 trials and 300 separate planting sites have been established across Australia.

This paper reports on the results of these trials under two different landscape conditions. The results are reviewed in the context of the opportunity for commercial plantations integrated with agriculture in low to medium rainfall areas to address salinity, carbon sequestration and sustainability. In addition, the potential for partnerships among land-rich, cash-poor farmers, catchment management authorities seeking to invest in public good projects that enhance environmental sustainability, and major carbon producing industries with either mandated or voluntary requirements to curtail net emissions will also be discussed.

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MATERIALS AND METHODS

Site type 1 – Shallow saline watertable with saline irrigation

In 1998, the first Saltgrow trial was established at Mt Scobie, near Kyabram in northern Victoria. It comprises 217 genotypes from two E. camaldulensis x E. grandis families and four E. camaldulensis x E. globulus families. This trial is a single tree plot, incomplete block design with five replicates and a single irrigation treatment using pumped saline groundwater.

The site forms part of a salinity control experiment testing conjunctive water use as a form of integrated on-farm salt management. The trial is located within the draw-down zone of a groundwater pump, and is used to dispose of the saline water extracted by the pump. Irrigation with low salinity water continues on surrounding dairy pasture. The trees are located in an area which had become too saline for continued pasture production, with a soil salinity of approximately 8 to 12 dS/m (ECe). The predominant soil type in the trial area is a Goulburn Loam (a grey-brown loam with a subsoil of yellowish-brown medium to heavy clay), with a tongue of Congupna Clay (grey, gilgai clay with a heavy clay subsoil) traversing the trial.

The site was irrigated with fresh water for five months following establishment. In the second irrigation season (October 1999 to March 2000), groundwater was diluted 1:1 with fresh water to achieve an EC of 5 dS/m. From the start of the third irrigation season (October 2000) and for all subsequent irrigation seasons, the site has been irrigated with undiluted groundwater at 10 dS/m. As a genetics trial, irrigation with saline water exposes all trees to the same level of salinity.

Mean annual rainfall in the area (Kyabram) is approximately 465 mm, and mean annual evaporation is about 1606 mm. Applied irrigation provides the equivalent of approximately 400 to 600 mm of rainfall, bringing the annual total to around 865 mm. Watertable depth across the site varies between 0.4 and 1.85 m.

Site type 2 – Medium to low rainfall, non-saline recharge sites

In August 2000, a series of six species trials, each comprising 84 hybrid clones and seven unimproved pure species judged as ‘best bets’ for low to medium rainfall areas, was established in conjunction with State Forests of NSW throughout the western slopes of NSW from Wagga in the south to Boggabri in the north. Pure species used in the winter rainfall trials were E. cladocalyx, E. camaldulensis, E. sideroxylon. Corymbia maculata and Acacia mearnsii. In the intermediate and summer rainfall trials, E. cladocalyx and C. maculata were replaced by E. argophloia and C. variegata respectively. The trials span the gradient from winter maximum rainfall, through even annual rainfall to summer maximum rainfall. Rainfall across the six sites ranges from 531 to 707 mm. Other key climatic data are summarised in Table 1.

These six trials were established on non-saline recharge areas, where rainfall not used by annual crops and pastures leaks past the rootzone. Over time, this leakage contributes to a rise in the watertable and leads to salinity outbreaks in downslope areas. The re-establishment of trees in recharge areas aims to prevent rainfall leaking to the watertable, preventing or limiting the spread of salinity.

RESULTS

Site type 1 – Shallow saline watertable with saline irrigation

Figure 1 illustrates the significant gains in stem volume achieved by the hybrids over their pure species parents in the Mt Scobie trial at six years (72 months). For the E. camaldulensis x E. grandis hybrid, the mean volume of all clones is 102 per cent greater than the volume of the best of the two pure species parents. Selection of the top ten per cent of clones increases this yield gain to 203 per cent. For the E. camaldulensis x E. globulus hybrid the volume gain is more pronounced, with the mean stem volume of all clones being 295 per cent greater than the volume of the best of the two pure species parents. Selection of the top ten per cent of clones increases this yield gain to 516 per cent.

Survival of the top ten per cent of E. camaldulensis x E. globulus and E. camaldulensis x E. grandis clones is 100 per cent and 95.6 per cent respectively, compared to pure E. globulus (five per cent) and pure E. grandis (24.6 per cent) (data not shown). In contrast to pure E. globulus and E. grandis, survival of pure E. camaldulensis is 100 per cent, confirming the salt tolerance of this species, but the growth rate, projected at an average to year ten of 3.7 m³/ha/yr, is well below the commercially viable threshold of around 15 m³/ha/yr.

Site type 2 – Medium to low rainfall, non-saline recharge sites

While the mean stem volume of both hybrid types was similar to that for the other hardwood species, when compared to the best performing pure hardwood species, (River Red Gum – E. camaldulensis), the top ten per cent of E. camaldulensis x E. globulus and E. camaldulensis x E. grandis clones performed 63 per cent to 67 per cent better respectively, while the top clone of each hybrid performed 64 to 82 per cent better.

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TABLE 1

Key climatic data for medium to low rainfall, non-saline recharge trials. Decile 1 rainfall means that there is a ten per cent chance that the annual rainfall will be at or below this figure.
DISCUSSION

Genetic improvement of Eucalyptus for commercial productivity in low/medium rainfall and saline site conditions

The exploitation of heterosis has been one of the major successes of plant breeding in the 20th century (Cooper and Merrill, 2000). The results of this work indicate the potential for a similar revolution in eucalypts for forestry in stressed environments. Through inter-specific hybridisation and the incumbent capacity to exploit and re-package the wealth of natural genetic variation available among inter-breeding eucalypt species, substantial improvements have been achieved in adaptation and productivity.

The Mt Scobie trial site presents soil conditions that would typically be considered stressful for growth of non-halophytic tree and crop species: shallow watertable (0.5 to 1.05 m) leading to problems with root zone aeration; moderately saline groundwater (7.5 to 8.5 dS/m) having direct osmotic and toxicity impacts; saline soil conditions at the soil surface (ECe 4.7 to 7.1 dS/m) and in the active rootzone area (ECe up to 12 dS/m at 0.5 m), again having osmotic and toxicity impacts; medium to heavy clay texture restricting root penetration, soil aeration and plant water availability; and high subsoil pH (7.5 to 8.8 at 0.5 m and below) leading to problems with nutrient availability. These site conditions are typical of many saline degraded areas of the Murray-Darling Basin, and the Shepparton Irrigation Region in particular, presenting conditions unsuitable for agricultural crops which are typically restricted to soils of less than 2 dS/m (Ghassemi et al, 1995).

Under the stressful conditions of the Mt Scobie site, significant heterosis is displayed by both hybrid types relative to their respective mid-parent means. For stem volume at 72 months, the gain in the mean performance over the better of the two pure species parents was between 102 per cent for the E camaldulensis x E grandis and 295 per cent for the E camaldulensis x E globulus hybrids. The practical implication of this result is that an increased level of timber production can be achieved in the hybrids under shallow saline watertable conditions compared to naturally occurring pure species.
actions required to address salinity. Integrating commercial forestry into agriculture – one of the key saline areas. These attributes make the hybrids a viable option for other commercial eucalypt species in low to medium rainfall and tolerance, and are able to achieve high rates of growth relative to pure species under low/medium rainfall conditions. The results of the NSW dryland salinity trials indicate that Saltgrow hybrids display a broad spectrum of stress conditions. The growth rate of the hybrids continues to follow the same trend as exhibited to date, then they might be expected to achieve a commercially attractive harvest yield when grown under poor quality site conditions.

Similarly, the NSW recharge site trials, spanning a rainfall range of 530 mm/yr to just over 700 mm/yr, present conditions typically considered below the usual rainfall threshold for commercial tree cropping in Australia (700 mm/yr). For forestry purposes, low rainfall in southern Australia is defined as <600 mm/yr; medium rainfall as 600 to 800 mm/yr, and high rainfall as >800 mm/yr (Harwood and Bush, 2002). Salinity in Australia predominately occurs in the 400 to 700 mm rainfall belt, and as such, species suitable for commercial reforestation of recharge areas of saline catchments need to be both tolerant of drought and water-use efficient to grow well in moisture limited conditions. The results of the NSW dryland salinity trials indicate that Saltgrow hybrids show improved performance relative to pure species under low/middle rainfall conditions.

Together, the Mt Scobie and NSW recharge trial results indicate that Saltgrow hybrids display a broad spectrum of stress tolerance, and are able to achieve high rates of growth relative to other commercial eucalypt species in low to medium rainfall and saline areas. These attributes make the hybrids a viable option for integrating commercial forestry into agriculture – one of the key actions required to address salinity.

**Carbon sequestration in planted forests**

Planted forests capture and store carbon from the atmosphere both directly in the woody and non-woody biomass (roots, branches, leaves and stem), and indirectly in the soil and forest litter. After harvesting, some carbon is released to the atmosphere, but modelling by the Australian Greenhouse Office indicates that much of the stored soil carbon is retained and increases over time though successive rotations. In addition, if timber is used in products such as house framing, then a proportion of stored carbon is locked out of the atmosphere for a longer period than the life of the forest itself.

Figure 4 illustrates the output from the carbon accounting model, CamFor, for a theoretical stand harvested on a rotation of 30 years, with periodic thinning, and achieving a mean annual increment of around 6 to 8 m³/ha/yr in each rotation. It can be seen from this figure that although the stand is harvested and replanted each 30 years, and that carbon stored in the trees and tree debris returns to zero at the end of each rotation, there is a continuing increase in both soil carbon and carbon stored in timber products.

Figure 5 illustrates the theoretical carbon profile for a eucalypt plantation estate planted at 1000 ha/yr up to a total area of 20 000 ha. The timber volume is assumed to grow at an average rate of 15 m³/ha/yr, with harvesting on a cycle of 20 years. It can be seen from Figure 5 that a plantation, even when harvested on a regular rotational cycle, creates a pool of stored carbon since, in any one year, an amount equal to only one/rotation age of the entire estate (in this instance 1/20th) is harvested and not actively sequestering carbon.

Figure 5 also shows that although the annual carbon sequestration for each hectare is around 34 tonnes of CO₂ and that rate of annual plantation establishment is flat at 1000 ha/yr, the profile of cumulative sequestration for the estate pool up to year 19 is geometric, being the sequestration from 1000 ha in any one year, the accumulation of year one plus that of 2000 ha in year two and so on.

**The scale of forestry required to address salinity and provide useful carbon sinks**

In contrast to high rainfall production forestry, forestry for the aim of both reduction of groundwater recharge and direct treatment of discharge sites will require the targeted re-introduction of trees as a mosaic in the rural landscape. However, for trees to exert an appreciable effect on regional groundwater tables, the sum of areas planted as a mosaic across the landscape must reach a scale rivaling Australia’s existing plantation resource in high rainfall areas. The scope of reforestation envisaged by the Murray Darling Basin Commission Salinity Reforestation Bank is in the order of
1.5 million hectares within the 500 to 800 mm rainfall zone, or just 4.2 per cent of the land area of this zone. This scale of reforestation, the associated scale of capital investment, and the trend of natural resource management agencies to leverage limited public funds with private investment, virtually dictates, in most situations, that such forests provide economic returns in order to encourage and sustain the scale of capital investment required.

Improvements in productivity achieved by Saltgrow hybrids under stressed environmental conditions should enable commercially viable growth rates to be achieved in the key areas requiring reforestation. Current projections are that Saltgrow hybrids can achieve harvestable logs of 40 to 50 cm diameter in 20 to 25 years. Further improvements in clonal selection may reduce this period, while developments in sawmilling and veneering technology may reduce optimum log sizes and rotation lengths.

Fortuitously, the need for large-scale reforestation to control salinity is fully consistent with the requirement for development of a critical resource mass to supply any industry based on processing of wood and fibre products. While hardwood sawmilling operations (currently based exclusively on native forests) can operate on a resource of as little as 5000 ha (RIRDQ, 1996), an internationally competitive softwood sawmill requires a minimum resource base in the order of 38 000 ha (RIRDQ, 1996). It is likely that a similar scale of resource will be required to support hardwood mills based predominately on new plantation timber resources, and that such mills will need to be integrated to produce a range of products that fully utilise the wood fibre entering the mill gate. Integrated timber processing operations would also utilise residues for products such as biomass energy generation, which in turn will contribute to meeting Australia’s renewable energy targets.

Similarly, large areas of plantation must be established to offset even a small proportion of CO₂ emissions from combustion of coal, or to provide offsets for export coal. For a project such as BHP Billiton’s Denrobium colliery yielding around one million tonnes of thermal coal/year, potential CO₂ emissions equate to around 2.75 million tonnes per year. For a plantation with a mean annual timber increment of 15 m³/ha/yr and a harvest rotation length of 20 years, the average annual carbon sequestration assuming a planting rate of 1000 ha/yr up to a total estate of 20 000 ha will be in the order of 337 000 tonnes of CO₂/ha/yr over the first 19 years. This equates to an offset of just over 12 per cent of potential annual CO₂ emissions from combustion of Denrobium thermal coal. While clearly, timber plantations

**FIG 4** - Cumulative carbon sequestered (mass/ha) as predicted by the carbon accounting model CamFor, for a theoretical plantation managed on a 30 year cycle for a mixture of solidwood and other timber products, with replanting following each harvest. The reduced rate of carbon sequestration in year 179 simulates the effect of a catastrophic fire in part of the plantation area (source: http://www.greenhouse.gov.au).

**FIG 5** - Cumulative carbon pool for a eucalypt forest under the following assumptions: average growth rate of stemwood = 15 m³/ha/yr; annual planting rate = 1000 ha/yr; harvest rotation = 20 years with each hectare replanted in the year following harvest.
alone cannot hope to achieve carbon neutral coal production, they can potentially make a significant contribution to offsetting carbon emissions within the scope of the reductions required by Kyoto restrictions. The significance of this lies in the realisation that timber plantations can begin to offer this benefit today, and continue to offer this benefit for the next 20 to 30 years until such time that all potential timber land resource is occupied. While the potential capacity for carbon sequestration in plantations is finite, the 20 to 30 year window of opportunity they provide may allow the time necessary for alternative technologies with much greater carbon abatement capacity, such as geo-sequestration and improved efficiency renewable energy, to be developed to a cost effective level.

A further benefit of large-scale plantations integrated with agriculture, in addition to providing salinity and carbon sequestration benefits together with commercial timber, will be the flow-on benefit to rural communities. Such plantations can be expected to create new regional industries and jobs in forestry contracting, timber harvesting, sawmilling and value added processing. This in turn will have flow-on benefits to service and support industries, and provide a diversified source of income to landholders, either through lease payments on land, or participating in the returns from tree growing. The net environmental and economic benefits for a single 20,000 ha resource catchment managed over a period of 40 years, has an estimated net present value in the magnitude of A$23 million (using a discount rate of eight per cent). This does not include the social benefit arising from the arrest of rural decline, stimulation of new regional jobs and industries, and the maintenance or improvement in farm productivity and sustainability.

Potential for partnerships between farmers, catchment management authorities and major carbon producing industries

The scale of reforestation required to both address salinity and provide useful carbon sinks will require significant capital investment. The Murray Darling Basin Commission have estimated an investment requirement of A$17 billion over the next 50 years to develop an estate of 1.5 million ha in the Murray-Darling Basin alone. Full funding for such investment from catchment boards, even where economic returns are attractive, is undoubtedly impractical. At the same time, the task is unlikely to be achieved by land-rich, cash-poor farmers, the custodians of the land on which re-introduction of trees is required, given the long investment periods to realise returns from forestry. This opens the opportunity for partnerships between catchment authorities, landholders and investors able to absorb the cash-flow profile of forestry investment, where most expenses are incurred at the start of a rotation and returns at the end. Given this nature of forestry investment, major carbon producing industries who can directly benefit from the annual accumulation of a carbon pool, who have long investment horizons, and operate at a scale with the capacity to absorb long periods of investment prior to realising a return, are ideally suited as investors in such projects.

From such a partnership, each party stands to gain significantly. Farmers can provide the land on which re-introduction of trees is required to achieve benefits to downstream land managers, water users and native ecosystems. At the same time farmers making land available for tree planting may benefit from lease fees, giving them certainty of annual income from a portion of their property, and from the direct on-site benefits of tree establishment including: shade and shelter for stock; windbreaks for crops, pastures and stock; habitat for natural predators - allowing reduced pesticide use; and direct return of degraded land to productive use.

Catchment management authorities, mandated with repair and improvement of the natural resources within their jurisdiction, benefit from the ability to leverage limited government funds with private investment in public good projects that enhance environmental sustainability, achieving a larger scale of land-use change than they could with public grant funds alone.

Finally, major carbon producing industries as investors receive long-term returns from: harvesting of timber products; the goodwill of facilitating public good environmental works; stimulating rural economies; and, carbon sequestration credits at a net profit over the life of the project. The carbon credits that can be generated from forestry investment can, in turn, be offset against emissions by purchasers, potentially providing a market advantage, particularly where purchasers are subject to carbon emission constraints.

CONCLUSIONS

Improvements in productivity achieved by Saltgrow hybrids under stressed environmental conditions should enable commercially viable growth rates to be achieved in the key areas requiring reforestation for salinity abatement. The scale of reforestation required to address salinity is consistent with the scale of plantation required to supply world competitive scale timber processing facilities and at the same time, this scale of plantation establishment can make a significant contribution within the scope of the reductions required by Kyoto restrictions to offsetting the potential carbon emissions from combustion of Australian mined coal. The significance of this capacity lies in the realisation that timber plantations can begin to offer carbon sequestration benefits today, and continue to offer this benefit for the next 20 to 30 years, providing a window of opportunity that may allow the time necessary for alternative technologies with much greater carbon abatement capacity to be developed to a cost effective level. Significant opportunity exists for major carbon dioxide emitting industries to partner in projects with salinity and other environmental benefits, including carbon offset credits at a net profit over the life of the project.

ACKNOWLEDGEMENTS

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Environmental Reclamation Practice in a Brazilian Coal Mine — An Economical Approach

J C Koppe¹, A Grigorieff² and J F Costa¹

ABSTRACT
Coal mining reclamation is a worldwide concern. This paper presents a Brazilian example where the economic aspects of reclamation are considered. The basic goal of the study is to transform the reclamation procedures into an economic process by integrating the use of land after reclamation and developing profitable post operations. Some of the activities considered in this case study include:

1. using of the coal pit as a landfill, considering appropriate landfill design and construction;
2. using the area for forest development; and
3. developing grassland, forests of native species and the construction of ponds for fishery.

This study will conclude that the mined area is:

1. capable of storing large quantities of waste at a competitive cost and at a low environmental risk;
2. commercial forest development is economically feasible; and
3. the lakes and other reclaimed areas can be integrated into the landscape creating a fresh water supply and an area to raise cattle.

INTRODUCTION
Coal mining has been occurring in Rio Grande do Sul since 1883. The previous surface coal mines usually disturbed large areas (Teixeira et al., 1996) and until 1980 little was done on coal mining reclamation practices. At that time, a conservative environmental law was introduced and mining companies started to develop their environmental programs. As a result, Brazilian coal companies have increased their understanding, awareness and expertise. They have particularly learnt from some very costly past environmental mistakes.

Economic factors have a strong influence on environmental decision-making. The sustainable supply of mineral commodities requires a balance between development, environmental, social and cultural objectives (Lambert, 1996). The development of profitable activities after the rehabilitation of the mined area is very important for the sustainability of mining operations.

To obtain high standards of environmental reclamation expected by the community, coal mining companies must perform beyond the levels determined by the imposed regulations. Consequently, the high costs associated with the reclamation process, particularly in impacted areas, has introduced the necessity of developing new practices in the field.

This paper presents examples of coal mine reclamation in Brazil, where economic factors are being taken into consideration. The basic goal is to transform the reclamation procedure into an economic process, integrating the final land use into the reclamation design to develop profitable post operations. The use of coal pits as landfill, reforestation and ponds for fishery are practices that will be discussed.

SITE LOCATION AND MINING ASPECTS
The coal mining site in this study is located 80 km west of Porto Alegre, Rio Grande do Sul, the capital city and one of the largest cities in the southernmost state of Brazil (Figure 1). Several coal beds are mined by strip mining using a back-hoe hydraulic excavator/truck system at the Recreio Coal Mine, one of the mining operations of Copelmi Mineração Ltd. The mine’s annual production is 1 800 000 tonnes at a stripping ratio of 7:1 (m³/t). The total material (coal + waste) excavated per annum exceeds 25 000 000 tonnes.
The strip mining starts with the removal of the top soil which is stored for rehabilitation of the area. Next, the overburden is removed by a mobile equipment fleet consisting of hydraulic backhoe excavators and trucks. Interburden is removed using a similar method to overburden removal. The waste fill is placed in the previous strip mined area and is dumped in accordance with the original stratigraphy, thus ‘recomposing’ of the original terrain. Coal is hauled to the processing plant where distinct products are obtained. Figure 2 shows the main operations during the coal mining process.

Several piezometers surrounding the mining area show that there is no problem with acid drainage or other local contamination of the groundwater.

**RECLAMATION PROCEDURES**

The Copelmi Environmental Program started in 1980 and comprises not only reclamation activities but also environmental education policies. This program is directed by the Brazilian and state government environmental laws and regulations and includes the involvement of local community and environmental agencies.

During the mine planning stage, all possible environmental impacts are taken into consideration. For example, delay between the stripping operation and reclamation is minimised, with continuous monitoring. Standard reclamation procedures are described in more detail below.

The mined area is ‘recomposed’ to the approximate original contour soon after the stripping process is completed. Next, the waste material is covered with the stockpiled topsoil. An erosion control system is developed as well as pH soil quality correction consisting of calcareous addition. Planting of perennial grasses takes place in an appropriate manner followed by reforestation. Monitoring and maintenance are carried out on a continuous basis. Figure 3 shows a general picture of the main reclamation procedures used.

**THE ECONOMICAL APPROACH**

The basic goal of Copelmi Ltd, considering the sustainability of the coal operation, is to transform the reclamation process and procedures into a viable economic process. The idea is to integrate the use of land after the reclamation process as a way to develop profitable post operations. Some of the activities are:

1. using the coal pit as a landfill,
2. using the area for forest development; and
3. the establishing of grassland, forests (involving native species) and ponds for fishery.

Finding appropriate areas for domestic waste disposal is a worldwide problem, especially in large cities. All countries worldwide are facing the solid waste disposal dilemma at various degrees. For instance, Porto Alegre (the capital of Rio Grande do Sul), located 80 km from the mine, has a daily solid waste generation of approximately 1800 tonnes and there is no available land nearby to construct a new municipal solid waste landfill (MSWL). Considering this situation, Copelmi Ltd proposed a large MSWL in the mined out pit of Recreio Coal Mine.

The deposition of domestic and industrial waste requires an appropriate landfill design and construction where certain geotechnical and hydrogeological constraints must be met, and both economical and environmental factors need to be considered (Koppe et al. 2002). The mined out open pit provides a large storage capacity and meets the required geotechnical and hydrogeological constraints for the installation of the MSWL. Given the area was initially impacted by mining activities, the regional environmental impact is potentially reduced considering that the excavation for a new MSWL in virgin areas close to urban regions will cause a worse impact. Figure 4 represents a cross-section of the MSWL and the main geological units involved. The MSWL underlain material consist of waste dumps and sedimentary rocks, some of these materials (waste dumps and sandstones) are permeable and need a liner to avoid solutions infiltration. The liner comprises 2.5 m of clay soil with very low permeability intercalated by 0.2 m of sand layer covered by a geomembrane and another 0.2 m of clay soil. The permitted capacity of the MSWL is 1000 tonnes per day of domestic waste and the operation commenced in September 2001. The MSWL construction cost approximately US$ 300 000. Presently, 92 municipalities are sending their domestic waste to the MSWL and the landfill is working at 80 per cent of its daily capacity.

Some of these towns are more than 200 km from the mine site. The average cost to store the waste including transportation is US$ 6.00/tonne for towns within an 80 km radius. Figure 5 shows the MSWL daily operation. The trucks transporting the domestic waste dump it into the MSWL, and a dozer helps to level and cover the waste with silty rock available in the mine site.

In some areas, where mining occurred during 1987 – 1991 the reclaimed areas were used for the development of demonstration forest projects. *Eucalyptus Sp* and *Acacia Sp* were selected for planting (Figure 6). The first commercial cut of these forests was carried out in 1999 and the wood was sold to a nearby paper mill. The planting of these species resulted in a very profitable operation, with an internal return rate of 50 per cent.
Other areas followed approved reclamation plans, by planting native grass species and plants for general landscape rehabilitation. These areas were returned to their former owners for cattle farming (Figure 7). A few ponds were also constructed as a source of water for animals.

In one specific area of the Recreio Coal Mine, a large pond was built and it is used during the dry season to supply fresh water for a nearby town. The quality of the water from this pond meets drinking water regulations. In the same pond an experimental fishery facility was also developed (Figure 8). Although this experimental project obtained good results the project was not continued, the local demand was not able to sustain the fish production.
CONCLUSION

Reclamation procedures viewed as an economic process, by integrating the use of land after reclamation, can lead to profitable post operations for mining companies, even if the economic factors were not taken into consideration during the feasibility studies or when the reclamation plan was developed. It is important to note that this is particularly beneficial to the sustainability of mining operations as well as achieving a balance between development, environmental, social and cultural objectives.

The study concludes that the mined area can:
1. economically be converted to a landfill at a competitive disposal cost and at a low environmental risk;
2. support commercial forests which represents a competitive business for the company; and
3. be integrated into the landscape by establishing grassland, native forests and ponds, producing grazing and water supply for cattle.

REFERENCES


Status of Outburst Research at the University of Wollongong

N Aziz¹, F Sereshki¹ and D Bruggemann¹

ABSTRACT

There has been an ongoing research on coal and gas outburst for the past two decades at the School of Civil, Mining and Environmental Engineering, University of Wollongong. Research study began with a humble beginning, initially conducting basic laboratory studies on the coal and gas properties, progressing into the determination of gas content of coal by sorption technique and the effect of gas pressures on coal strength. The present laboratory facilities and research interests are extended to include the study of coal permeability and shrinkage properties and their effect of gas drainage characteristics with use built gas type, and pressures. All the changes are examined with respect to changing in-site geological conditions of the coal deposit investigated. The aim is to provide a long-term support to industry in establishing a data bank for Australian coal deposit characteristics and properties.

INTRODUCTION

For more than two decades, there has been a continuous program of research at the School of Civil, Mining and Environmental Engineering, University of Wollongong. Much of the early research studies were carried out in collaboration with the late Dr Ripu Lama. Initially the main study was related to sorption technique for determining gravimetrically the gas content of coal, and the extended later to volumetric method. Other studies undertaken included the modelling of gas sorption in coal (Nguyen, 1988). The next phase of the research involved the development of a multi function outburst rig (MFORR) for outburst research. The MFORR was initially used to study the effect of gas environment on the strength properties of coal including:

1. The effect of gas pressure on coal tensile strength, using the well known Brazilian method of indirect tensile testing of cylindrical core samples in different gas pressure confinements.
2. The effect of gas pressure gradient on coal load bearing capacity.
3. Study of the strength of coal by examining the particle size distribution of drill cuttings under different gas environments. A high precision drill of controlled speed up to ten different levels was used to study the changes in particle size distribution with respect to increased gas type, gas pressure. The changes in coal strength properties were also compared with drilling of coal in air (Aziz, Hutton and Indraratna, 1996).

Concurrent with the above, an extensive study of various coal seams gas content was conducted using an in-house built adsorption and desorption equipment used previously for indirect method of determining the gas content of coal. The only modification introduces to the bomb is the addition of pressure transducer on the lid of each bomb to monitor the bombs inlet gas pressures. Coal samples are sealed in gas bombs and pressurised to a saturation level at 3 MPa. It is then immersed in a water bath to maintain it at a constant temperature of around 25°C. A thermostatically controlled water bath (with a stirrer) allows the coal samples to be kept at the desired temperatures. Further details of equipment construction, operation and gas content calculations at various pressure levels are described elsewhere (Aziz and Ming-Li, 1999).

EQUIPMENT DESCRIPTION

Adsorption and desorption apparatus

This equipment has been the focus of outburst program research for the past two decades. Initially it was constructed to determine indirectly, and gravimetrically the gas content of coal at different gas pressures, nowadays it is also used for coal sample preconditioning, prior to permeability, coal shrinkage and coal strength tests. The apparatus (Figure 1) consists of number cylindrical pressure vessels, known as pressure ‘bombs’. Coal samples are sealed in gas bombs and pressurised to a saturation level at various predetermined pressures up to 5 MPa. The sample containers are immersed in a water bath, but are isolated from the water bath by copper sleeves to keep them dry. A thermostatically controlled water bath (with a stirrer) allows the coal samples to be kept at the desired temperatures. Further details of equipment construction, operation and gas content calculations at various pressure levels are described elsewhere (Aziz and Ming-Li, 1999).

Coal shrinkage test

Figure 2, is basically the pressure vessel (bomb) component of adsorption and desorption equipment used previously for indirect method of determining the gas content of coal. The only modification introduces to the bomb is the addition of pressure transducer on the lid of each bomb to monitor the bombs inlet gas pressures. Coal samples are sealed in gas bombs and pressurised to a saturation level at 3 MPa. It is then immersed in a water bath to maintain it at a constant temperature of around 25°C.

Before, the coal samples are placed in the bombs; four strain gauges are mounted on each sample surface to monitor axial and radial strains on coal size due to gas sorption. The mounting of the strain gauges is carried out in accordance to International Society of Rock Mechanics (ISRM) standard. A data taker ‘model DT50’ is used to retrieve information from the bomb which is then connected to a PC for data analysing.

Multi function outburst research rig (MFORR)

MFORR comprises a number of components, which can be utilised on a variety of research studies, initially built for the study of the evaluation of changing coal strength properties with respect to changing gas environment of the coal sample tested. At present the rig is used mainly for coal permeability studies. The integrated components of the MFORR include:

1. School of Civil, Mining and Environmental Engineering, University of Wollongong, Wollongong NSW 2500.
1. main frame,
2. gas pressure chamber – also used for coal permeability studies,
3. drilling system,
4. drill support frame,
5. drill cutting collection system,
6. universal socket for vertical load application,
7. flow metres (see Figure 4),
8. data acquisition system, and
9. various components for coal strength properties tests.

When used as a precision drill, the pressure drill rig (PDR) consists of drill frame, drill motor with drill bit, drilling thrust system and drilling cutting collection device. A multi-pulley system enabled constant thrust to be applied on the drill bit. The thrust is generated by a suspended steel cylindrical bucket filled with lead shot. The drill cuttings are collected in a specially designed catcher, fitted with a disc of filter, and connected to a suction pump. The collected drill cuttings are subsequently weighed and analysed for particle size characterisation. A Malvern particle size analyser is used to conduct particle size analysis of drill cuttings. The particle size analyser is capable to classifying particle sizes between 1 mm and 0.5 µm.

**MFORR for permeability test**

When MFORR is used for coal permeability, the precision drill section and drill cutting collection system are disengaged and the gas pressure chamber is reassembled to cater for the needs of the permeability tests. Figure 4 shows the schematic diagram of the test rig (Aziz, Porter and Sereshki, 2004). The high-pressure gas chamber is connected to a set of flow metres for monitoring gas flow rates. To conduct the test, the samples are cut into 50 mm lengths, and the ends polished. In the centre of each sample, a 6 mm hole was drilled through each sample. The sample ends are then sealed with a lock-tite seal. The core sample is then placed between loading plates of the chamber. Axial strain is then applied to the core sample via a universal torque. Changes in the sample axial and lateral load dimensions due to gas sorption are monitored by two sets of strain gauges. Parameters that are monitored include:

- application of stress,
- measurement of strain on the sample,
- measurement of gas flow rate,
- application of constant circumferential gas pressure, and
- application of constant suction.

Gas is charged into the sealed pressure chamber at a pressure of 3 MPa and maintained constant for a period of one week to allow the coal to be sufficiently saturated. The strain is recorded for this period. In the tests reported here little change in strain was observed over the time period. Once the sample was fully saturated, the release valve was opened and released gas passed through various flow metres of differing flow rates consisting of:

- low flow range: 0 - 100 ml/minute,
- medium flow range: 0 - 2 L/minute, and
- high flow rate: 0 - 15 L/minute.

Information from the load cells, strain gauges and flow metres were monitored in a data logger connected to a PC.

**RESULTS AND DISCUSSION**

**Gas type and pressure and coal strength relationship**

Figure 5 shows the bar charts of three different gas sorption quantities in Bulli coal seam, Sydney Basin. The gases used were CH₄, CO₂ and CH₄/CO₂ (50 per cent) mixture. There is a clear trend of different gas sorption quantities in coal, with the higher sorption being of CO₂ gas.

Figure 6 shows the average values of drill speed record of coal specimens tested under both in air (ie normal atmospheric condition) and under increased gas pressures of 1500 and 3000 kPa. Ten tests were made for each sample environment. The rate of drilling of coal samples in air was relatively slower than that drilled in higher confined gas pressures. The highest values were obtained in CO₂ confinement. The increase in gas pressure to 3000 kPa also resulted in an increase in the rate of drilling.
Figure 7 shows particle size distribution of drilling cuttings in various gas pressures. The graphs represent the mean line for ten samples tested under each gas type and pressure. The particle size distribution ranged between 0.5 µm and 878.67 µm. Drilling in air produced finer particle sizes than drilling under gas pressure confinement. Additional observations made include:

- Drilling in CO₂ environment produced coarser particle sizes than in CH₄ and CH₄/CO₂ environment at 1500 kPa pressures.
- The coarse particle size were lower in CH₄/CO₂ and even lower in CH₄ alone environment.
- Increasing CH₄ gas pressure confinement to 3000 kPa produced coarser drill cuttings. In fact the particle size distribution for CH₄ at 3000 kPa was similar to that produced from drilling in coal saturated with CO₂ gas at a confinement pressure of 1500 kPa. This is to be expected, as the increased gas pressure to 3000 kPa may have forced more gas into coal micropores leading to a reduction in surface energy of the coal.

All this indicates that the presence of confining pressure has a detrimental effect on the strength of coal. It is possible that the presence of sorbed gases in coal at higher pressures may weaken the coal tensile strength by introducing micro-fractures into the coal structure. According to established facts and reported by Gray (1995), heavily fractured and soft rocks usually produce coarse drill cuttings with high rate of drill penetration.

**Coal shrinkage test results**

Changes in the volume of coal matrix were calculated using the average of the two strains in the axial and radial directions. The shrinkage coefficient ($C_w$), is defined as the rate of change of coal matrix volume to the change in gas pressure and is given by (Harpalani and Chen, 1997):

$$C_w = \frac{1}{V_m} \left( \frac{dV_m}{dP} \right)$$

where:

- $V_m$ = matrix volume (m³)
- $dV_m$ = change in volume (m³)
- $dP$ = change in applied pressure (MPa)
- $C_w$ = shrinkage coefficient (MPa⁻¹)
Figure 8 shows the relationship between applied gas pressure and volumetric change in coal. The coal sample was initially charged to a maximum pressure of 3 MPa. The changes in coal volume were monitored in increments of 0.5 MPa. As can be seen, the reduction in coal volume is different for different gas medium. A minimal change in coal volume was measured with nitrogen while a CO₂ environment produced the highest volume change. Obviously, the influence of CO₂ reflects a strong affinity of the gas for coal. As coal adsorbs CO₂ more strongly than methane, it is thus likely the high rate of gas storage in coal is accommodated with the increase in coal volume. Clearly the change in coal volume in this case is more than five fold in CO₂ in comparison with the methane environment. The relative change in coal volume in mixed CO₂/CH₄ environment is between pure CH₄ and CO₂, but the mixture proportions influenced the degree of volume change.

**Coal permeability test**

Figures 9 and 10 are permeability graphs of coal samples tested in both methane and carbon dioxide gases under different gas pressures. The axial applied load was maintained constant at 2000 kg. The Bulli seam coal samples tested were collected from two geologically different locations in a local mine working Bulli seam in the Illawarra Coalfield of Sydney basin, NSW. Samples collected came from 800 panel (sample #800051) and 900 panel (sample #900114 and #900104). The geology of these two areas at hand specimen scale is significantly different and can be described as:

1. 800 panel – ‘normal’ coal in terms of cleat spacing and orientation, orthogonal, regular spacing, normal ordered horizontal bright and dull layers, does not display visible deformation.
2. 900 panel – ‘structured’ coal with broken structure, cleats often not subvertical, cleat spacing irregular, occasional small scale dislocation amongst bright and dull layers. Calcite mineralisation often found towards top of seam, usually oblique to bedding plane but tends towards bedding plane in lower parts of each vein.

From a practical perspective, gas drainage has been exceedingly difficult in the 900-panel area when compared to the 800-panel area. Management has resorted to the ‘grunching’ method of heading development using explosives, particularly where gas content levels have been greater than the allowable gas threshold limits. The coal structure has been disturbed to a point where the contained gas does not freely move from high inseam fluid pressures to the drainage lines.

The permeability of each sample was calculated using the following Darcy flow equation (Lama, 1995):

\[
K = \frac{\mu Q l n (r_i / r_o)}{\pi l (P_o - P_u)}
\]

where:
- \(K\) = permeability (Darcy)
- \(l\) = height of sample (cm)
- \(Q\) = rate of flow of gas (cc/sec)
- \(P_o\) = absolute pressure in chamber (bars)
- \(P_u\) = absolute pressure in outlet (bars)
- \(r_o\) = external radius of sample (cm)
- \(r_i\) = internal radius of sample (cm)
- \(\mu\) = viscosity of gas

The results showed a marked difference in the resultant permeability between the 800 and 900 panel coals. The difference in permeability (in millidarcy) between 800 panel and the 900 panel coal for each of carbon dioxide and methane is quite different. 800 panel had approximately three times greater permeability when compared to the 900 panel coals (Figures 9 and 10).

Permeability tests for both carbon dioxide and methane show that the 900 panel coals have much lower permeabilities than the 800 panel coals. Since permeability is a function of a number of parameters including size, distribution and frequency of cleats, any phenomenon that reduces cleat porosity will decrease permeability. Given that 900 panel coals contain much higher carbonate contents than the 800 panel coals, and also have the lowest permeability, it is suggested that the reduced porosity of the 900 panel coals is due to the infilling of the cleats with carbonate.

The reduced permeability value explains why the 900 panel area is much harder to degas. The carbonate in-filled cleats restrict the movement of gases from the surrounding coal to the gas drainage holes.

![FIG 9 - Coal permeability in carbon dioxide at different gas pressures and at 2000 kg axial load.](image)

![FIG 10 - Coal permeability in methane at different gas pressures and at 2000 kg axial load.](image)
Gas flow modelling

A preliminary computational fluid dynamics (CFD) modelling exercise was carried out to 'visualise' the gas flow in the porous coal sample. Figure 11 shows the computational domain and the corresponding computational mesh used for the simulation. A thin slice of the coal sample was chosen as the computational domain, in order to take advantage of the axial symmetry of the sample geometry.

The computational domain is divided into a number of non-overlapping subdomains called 'cells'. Equations describing the conservation of mass, and the Darcy equation, which replaces the momentum equations in the fluid mechanics of porous media (Bejan, 1984), are solved iteratively until balances are achieved for each computational cell. Since the cells are contiguous, this implies balances for the entire computational domain. Results are presented in the form of velocity vector plots and pressure contour plots. A typical experimental condition was chosen for the simulation. For this flow, a typical permeability value of 1 mD was used (Figure 12b). A stagnation pressure condition was applied at the inlet, and a zero gauge pressure at the outlets. For the flowing gas, CO₂, the following relevant properties at 300 K were used (Incropera and DeWitt, 1996):

- **Dynamic viscosity** = 149 (10⁻⁷) N-s/m²
- **Density** = 1.7730 kg/m³

The inlet and outlet conditions were:
- **Inlet stagnation (total) pressure** = \(5 \times 10^5\) Pa (gauge)
- **Outlet pressure** = 0 Pa (gauge)

Figures 12a and b show the results in terms of the pressure contours through the coal sample, and the flow streamlines, respectively. The results suggest that for the above flow geometry (radially impressed flow with high stagnation pressure values, through a small axisymmetric sample with a small centrally located outlet), some of the gas exiting through the vertical sides of the central hole finds its way out from the top, while some may be trapped in the lower part of the hole. Also, it is very likely that the gas flow, under the above experimental conditions, reaches extremely high velocities as it flows through the tiny fissures and cracks in the coal sample.

**OUTBURST WEBSITE**

ACARP is providing funds for the establishment of a website on coal/gas outburst.

The primary objective of this project is to develop an on-line coalmine outburst information management system to provide the coal mining industry with all the necessary information on outbursts via the world wide web. Such a system should provide easy access to the experiences acquired by the coal mining industry. Some of the attributes of such a system must include:
mechanisms of outburst;
outburst predictions and prevention;
current management controls and compliance;
relationships between geological structures and outburst events;
in-seam drilling techniques;
a virtual library of coalmine outburst with current and past literature and references;
hyperlink to proceedings of the current and past South Coast Outburst Seminars regularly organised by outburst research committees in Illawarra;
hyperlinks with other international websites to provide additional source information;
ease of use and accessibility; and
regular maintenance with current issues.

In summary, the proposed online system will electronically disseminate information on outburst for the Australian coal mining industry. It will dynamically manage both historical and current experiences of Australian and worldwide on coal mine outburst. The website (http://cedir.uow.edu.au/Projects/outburst) is currently at its infancy stage and is presently been upgraded gradually with new material in the coming months. The outburst website will be linked to the well known The University of Wollongong website on longwall mining (www.uow.edu.au/eng/current/longwall).

CONCLUSION

The program of research activities reported in this paper is a clear demonstration of our commitment in maintaining research on coal and gas outburst as a priority research for the benefit of the coal industry. It has been demonstrated that:

1. The study of the effect of gas pressure on coal strength through the analysis of particle sizes is a valid approach.
2. Permeability and shrinkage studies can serve as an effective approach in understanding the drainage characteristics of coal seam with intrusions and other geological disturbances. The effectiveness of these methods can be better enhanced through assessment of coal composition and mineralisation, which is the currently been enhanced.
3. The status of current research program perused at the University of Wollongong, is a continuation of the research work dating back to more than two decades. We are looking ahead to better utilise the latest know-how and technologies for the establishment of a predictive indices for effective coal deposit mineability.

4. The establishment of an ACARP funded new website on gas and outburst management (http://cedir.uow.edu.au/Projects/outburst), should serve as a useful platform for disseminating the latest findings in outburst control technologies, leading to safe and efficient mining in Australia and worldwide.

REFERENCES


The Demonstration of Electronic Systems to Assist in the Management of a Significant Incident

D Cliff¹ and R Moreby²

ABSTRACT

This paper is a demonstration of the techniques developed during an Australian Coal Association Research Program (ACARP) funded research project aimed to develop aids, principally computer based, to assist mine personnel to more efficiently manage significant incidents in coal mines. These aids include software to aid in the location of the incident, electronic information storage and retrieval, online site emergency response manuals and software to aid in decision-making.

INTRODUCTION

One of the recommendations of the inquiry into the 1994 Moura No 2 underground coal mine disaster, where 11 miners lost their lives, was that an emergency response exercise be conducted at an underground coal mine each year to test the mine’s internal emergency response system. It also aims to test the response of the Queensland Mines Rescue Service and other external agencies’ ability to respond and render assistance to the mine. Eight such exercises have been completed. In addition to this level one exercise each mine is required to carry out emergency exercises based on operating sections (level 3) and whole of mine (level 2) annually. These exercises have led to significant improvements in the way mines prepare for emergencies and in their abilities to manage the incidents. The findings of the exercises have been reported elsewhere (Rowan et al, 1998 - 2002, Reece et al, 2004, 2005).

The scenarios were tailored to conditions and situations that had occurred at the mines, eg roof falls, frictional ignitions, etc in addition to including historical incidents.

Many valuable lessons have been learnt from these exercises including:

- the move to compressed air breathing apparatus for in seam rescue and response;
- the use of sophisticated computer fire simulation programs;
- the use of life lines;
- the use of visibility impairment aids;
- the use of refuge bays/fresh air bases (FAB); and
- increased awareness/familiarity with using self-contained rescuers.

In general one of the areas that is dealt with least effectively is that of information management, ie collection, analysis and reporting. The information management issues relate to a number of key areas.

Mine environment monitoring systems

1. Despite the fact that all mines tested had sophisticated computer based communications and gas monitoring systems, much of the communication was done verbally and transmitted via handwritten notes. This led to a number of significant delays in obtaining appropriate information and on occasion incorrect information was obtained. Often the gas monitoring data was only displayed in the control room. The full capabilities of the gas monitoring software and computer system were not used.

2. No one person had the responsibility for obtaining and analysing ventilation and gas concentration data. Thus no one had responsibility for ensuring the quality of the data. Often key decisions were made without any understanding of the limitations of the data being used as the basis for those decisions.

Information flow/record keeping

3. Often there was no accurate information flow not just gas information but also relating to vehicle and personnel movements and locations. No effective recording procedures or logs of actions and decisions with reasons or evidence supporting those decisions.

4. There was often limited or no control over communications into and out of the incident management room. Often the briefings of incident management team (IMT) personnel were unstructured.

5. On a number of occasions there was ineffective communication of information to mines rescue superintendent and to rescue teams and other key personnel. The integration of rescue team organisational issues into IMT decision-making may have provided improved rescue effectiveness.

6. Inaccurate recording of persons underground and movement and location of persons underground. There was a lack of formal method to record and update the status and deployment of resources for rescue operations.

7. On most occasions important incident management decisions were not made until the mine manager could arrive on site – in some cases causing delays of over two hours. Information was transmitted to the senior official offsite by phone. There were several instances of incorrect information being received by the mine manager because of this.

Incident management room

8. In a number of exercises the Incident control room was poorly resourced, no provision of white boards, accurate mine plans, desktop space, communications facilities and security against intrusion.

Decision-making

9. All too often there was no record kept of the decision-making process.

10. Decision-making occurred over too long a time period, there was no sense of urgency, direction or focus – best provided by a clearly stated (and written up) set of goals, objectives and priorities.

11. On more than one occasion there was the development of a group think mentality for decision-making.

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DEVELOPMENTS

The development of tools to address a number of these issues has been progressed through an Australian Coal Association Research Program funded project looking at significant incident, investigation, evaluation and analysis (Cliff, Moreby and Meadowcroft, 2003) and has been presented previously (Cliff, 2003, 2004).

Using a recent emergency simulation exercise as the basis this paper will outline how these techniques may be applied. The scenario generated for the simulation was essentially a frictional ignition in a development heading. This caused a methane explosion that picks up some coal dust and develops sufficient energy to destroy some outbye overcasts – short-circuiting the ventilation to some of the longwall blocks being developed. In addition the shock wave from the explosion dislodges a methane drainage line and the following ignites the methane emanating from it as well as a small pile of coal dust at the drive head of the development heading. Neither fire is very big. Personnel from the area where the blast occurred cannot be contacted. Other personnel escape the mine and provide details of the fires and apparent explosion damage. The aim of the exercise was a search and rescue effort for the lost personnel and a plan for the stabilisation and recovery of the mine.

Technical information

At many times during the IMT discussion process ready access to key information is required. For example, when discussing whether or not personnel can go underground safely the Mines Rescue Guidelines should be consulted. An easy way for this to occur is to have them in electronic form available on a desktop computer/laptop or on a Pocket PC. E-books are a simple mechanism of converting text documents into easily readable electronic books with large fonts for easy reading and tables of contents and hot links for quick access to key areas. Microsoft® provide MS Reader® as a free program to convert MS Word® document to e-book format and to read e-books. This project utilised this program and another, ReaderWorks® to create templates that could be imported into Word to allow seamless conversion of Word documents into e-books. The user needs no formal training to use the template merely following the instructions at each stage. This template is provided on the CD with the final report available from ACARP.

Figure 1 outlines how this information would be displayed on either the computer or the pocket computer, explaining how key information can be calculated. The second part of the figure displays key heat/humidity information from the guidelines that defines how long rescue teams may operate in breathing apparatus.

The electronic display works equally well for mine site policies and procedures. Figure 2 outlines an example of a mine’s trigger action response plan for spontaneous combustion. A Pocket PC has touch screens to facilitate going between pages. In the example shown in Figure 2, progressing from page 9 to the details of what constitutes a level 3 trigger is easily achieved by touching the level 3 icon on the page.

Electronic books can be used on site computers, pocket computers and available offsite through the internet. These electronic books provide access to site procedures, response plans, trigger points, expert assistant databases and contact lists. The electronic documents permit good version control and restrict modification. For example a mine manager can be anywhere in the world and receive a phone call relating to an incident at their mine and be able to access all the key documentation instantaneously via their Pocket PC or laptop.
computer. As part of the ACARP project the mines rescue guidelines for NSW and Queensland were converted to e-book format as were the Queensland Coal Mining Safety and Health regulations, and a gas analysis and interpretation primer was prepared.

Personal pocket computers have been utilised to increase the portability of data communication and speed up the incident initiation process. As they are compatible with normal full sized computers they carry MS Outlook® giving access to address books, email and the internet.

In addition as they carry significant processing power (Pentium III processors) they can be used to carry out key calculations such as explosibility analysis. Using a simple easy to use and cheap (<$150) software package Pocket PC Creations, programs were written for the ACARP project to calculate the various standard gas indicators and explosibility parameters. This is demonstrated in Figure 3. Other programs were compiled to undertake safety surveys and collect information relating to manual handling risks.

**Display of key mine information**

Simple computer software (generic) has been developed to allow quick and simple acquisition of key mine environmental monitoring information, again accessible over the net, both intra and inter. This software has been developed in MS Excel® so that it is not subject to proprietary concerns or huge costs. This software can also be used to track the movements of vehicles and persons underground.

The data can be input directly into one of the input datasheets or connected via dynamic data exchange to mine monitoring systems where this is enabled. Data can be linked to other spreadsheets that might already exist, eg including gas chromatograph data.

Figure 4 shows one of the data entry screens.

Once the data have been entered there are a series of macro commands embedded in the spreadsheet that create the graphics boxes. The user inputs a mine plan graphic and then places the created textboxes as required. The graphics are initiated via a set of control buttons, which lead the user through the setup process.

As the program is Excel it has all the features of MS Paint® attached. For the scenario described in Figure 4, the diagram depicted in Figure 5 identifies the overcasts that were identified as being damaged with a red D. This allows people to quickly recognise the ventilation system has been compromised and consider the ramifications of this damage. In addition it is quickly apparent that a number of the fixed monitoring points are damaged (negative values or fresh air where it cannot be) and the information that they report is of no value. This mine has a number of real time ventilation sensors and they provided
**FIG 4 - Example of a data entry screen for display program.**

**FIG 5 - Example of the output available from the display program.**
valuable information to the IMT personnel in relation to the state of the ventilation circuits throughout the mine. The sensors indicate that there is air flowing in TG23 but not much in MG24 or MG25.

A library of symbols has been created to allow quick marking of key features such as fresh air bases and fire locations. This allows the mines rescue function to be planned out including search routes, fresh air bases, etc. Hard copy of the plan can then be given to the rescue teams to use as it already carries all the latest information. Figure 6 outlines a mine plan updated to include the mines rescue projected routes from the FAB marked. In addition the faulty gas detectors have been removed from the plan to ensure that only accurate information is known. This highlights the absence of any accurate gas data from the region where the incident occurred and the need to be careful.

As it is in Excel it can be linked to other spreadsheets or the user can create calculations and graphics as they desire. The program has been configured to create Coward Triangle explosibility analysis directly and this can be overlaid on the mine plan or plotted on a separate sheet as desired.

All types of data can be input including mines rescue information. There is a data entry screen set up to log all rescue teams, their members, function and time under BA. These can then be plotted onto the mine plans to show progress.

Now that the information is electronic it can be shared across the site and offsite as well. This also permits quick and accurate briefings of key stakeholders such as mines rescue teams, and mines inspectors. The diagrams can be printed out to give the rescue teams accurate information.

This technology can reduce the delay in responding to an incident, which is crucial in saving lives and preventing incidents worsening.

**Event logging and record-keeping**

Another use for Excel is to log key IMT information – actions, decisions and events. A feature of Excel is its auto filter function. As can be seen from the example in Figure 7 it can be used to quickly sort through the event log to identify one type of information, it could relate to current tasks, or a particular functional area such as inertisation. Again being electronic it can be shared with others and used to bring stakeholders up to date without disturbing the IMT process.

**Decision-making assistance**

Finally, the ACARP project addressed the issue of decision-making. A number of free or low cost decision-making and logic tree programs have been trialled to enable more effective decision-making to be undertaken. Decision trees have been developed for a number of scenarios. The software tracks the decisions made and logs the rationale behind each decision. The example in Figure 8, using the program Reasonable probes the question of whether or not it is safe to seal a panel.

It establishes a framework to assist the IMT in identifying and considering all the important factors and also flags information gaps.

There are many programs readily available that facilitate decision-making through documenting brainstorming processes. These are easy to use and based on a graphical user interface. The example in Figure 9 using MindManager® demonstrates how an issue can be explored very quickly. Hyperlinks can be included to attach other files to branches of the diagram. These could provide additional information or be action plans to follow if that branch of the decision tree was activated. These programs come with all the graphics support expected of modern computer programs and tutorials are provided to assist in obtaining proficiency. These


**Fig 7** - An example of the event log using MS Excel®.

**Fig 8** - Reasonable example of decision assistance.
diagrams provide a log of the thought processes; they can be stored for future reference or modified as the situation decrees. It is the intention of the current ACARP project to provide a number of decision-making templates preformed to allow minimal loss of time in a real situation. In addition under the pressures of a real incident, dispassionate logical thought is often not possible so developing rigorous evaluations of situations and thorough risk assessments is unlikely.

Increased use of computers allows sharing of information between a number of locations both on site and anywhere around the world. This in turn facilitates briefings of key groups such as mines rescue or government mine inspectors without disrupting the IMT. Information can also be entered from these areas without disrupting the IMT, which then optimises the decision-making process.

The decision-making process has deliberately been left in the control of the IMT. The electronic devices merely providing aids to improve the quality and the speed of the decision-making process.

CONCLUSIONS

In general the information display and transfer processes observed during level 1 emergency simulation exercises were found to be inadequate. This was in part due to the fact that many of the systems and roles had not been trialled fully to evaluate their practicality and value. It is imperative that during and incident the incident management team is able to obtain accurate adequate and current information to enable it to function properly.
In the 21st century it is appropriate to use 21st century techniques to assist in this. The computer-based systems outlined above are examples of how this can be achieved. These systems are examples only and no doubt better systems can be developed, tailored to an individual sites needs. It is imperative that such systems are established during normal mine operation so that in an emergency they can be utilised without any fuss or dislocation of effort. All software outlined above is either freely available in the final report or can be obtained from the Australian Coal Association Research Program via their website, http://www.acarp.com.au for a nominal cost or is shareware available for less than $200.

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Real-Time Risk Analysis and Hazard Management

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ABSTRACT
Safety remains a critical priority for the Australian mineral resource industry and will receive increased focus in the future. This is particularly evident in underground coal mines where reserves are becoming deeper and more hazardous to extract. The CSIRO, through its Exploration and Mining Division, has recently delivered on two projects aimed at providing step-change capabilities in real-time risk management and hazard control. This paper describes the key outcomes of these projects.

INTRODUCTION
The Australian Coal Association Research Program (ACARP) funded Location And Monitoring for Personnel Safety (LAMPS) Further Development project has completed the development of a system for the real-time location of personnel throughout the workings of a mine. The system includes:

1. an IEC EX ia (intrinsically safe) transmit-only tag that possesses an internal battery; and
2. an IEC Ex ia (intrinsically safe) tag reader for installation in explosive risk zones (ERZ0 or ERZ1).

The internationally funded Enhanced Mine Communication and Information Systems for Real-time Risk Analysis project offers unique capabilities for the real-time monitoring and management of hazards in our mines. The project, funded by the Japan Coal Energy Center (JCoal), ACARP and the CSIRO has been installed at Angle Coal’s Grasstree mine and is currently undergoing extensive field-trials.

The system, known as Nexys™ consists of both software applications and IEC Ex ia (intrinsically safe) hardware that provides for:

1. the real-time sourcing and integration of critical data sets from the range of propriety systems already in place at a mine, including the location of personnel and equipment, gas monitoring, strata, ventilation and SCADA systems;
2. analysis through a rules-based inference engine, development of 3D trigger action response plans, historical analysis and action logs; and
3. state-of-the-art 3D graphic interfaces and autonomous call-up the latest mine plans and current workings.

The system utilises fully managed, ethernet based communication protocols over multi- or single-mode fibre optic cables allowing for the future integration of the ever increasing array of ethernet enabled device (eg PDAs, VoIP phones, web-cameras, wireless ethernet sensors, virtual environments and base stations, training and emergency response technologies).

RECENT DEVELOPMENTS IN LOCATION MONITORING

Overview of LAMPS developments

The LAMPS system was developed by CSIRO and supported by two rounds of ACARP funding (see http://www.acarp.com.au/). The project commercial partner, MineCom Australia Pty Ltd, guided the developments.

The project outcomes include the development of the intrinsically safe LAMPS type II tag and the intrinsically safe LAMPS type I reader. LAMPS is the first system in which both tags and readers have been certified to the current international IEC standards for use in hazardous regions within underground coal mines.

Patenting action was carried out in the USA, Canada and Australia to protect the LAMPS innovations. The USA patent, No 6 339 709 (formerly Application No 09/448,898), in the name of CSIRO, was granted 15 January 2002, and its potential expiry date is 9 April 2018. The Australian patent, No 753 168 (formerly Application No 6 814 898), in the name of CSIRO, was granted 9 April 1998 and has a potential term of 20 years. The Canadian Patent, No 2 289 752 (formerly Application No 2 289 752), in the name of CSIRO, was granted 3 August 2004 and its potential expiry date is 9 April 2018.

The intrinsically safe LAMPS type II tag

ACARP, mine personnel and MineCom have independently suggested the development of standalone (or self-powered) active tags. A transmit-only expendable tag, known as the LAMPS type II tag, has been developed. It is powered by an internal lithium battery and the entire tag is completely encapsulated in potting compound. The tag has been designed to meet the international intrinsic safety standards, namely AS/NZS/IEC 60079 Part 0 and Part 11 for Gas Group I.

The LAMPS tag II transmits a packet approximately every three seconds and has a lifetime of at least two years. A packet re-transmit time of three seconds was selected to permit detection of underground mine personnel travelling in vehicles at (say) 10 km/h = 2.8 m/s. Suppose that readers can intercept tag packets at a minimum range of ±5 m, then the minimum packet capture envelope for a vehicle travelling at 10 km/h will be 10 m/2.8 m/s = 3.6 seconds: thus a re-transmit time of three seconds should be adequate. If the reader is mounted on the roof above the centre of the drive then underground mine reader ranges of at least ±50 m can be expected. This provides a design safety factor of ten, in order to accommodate faster vehicle speeds, packet collisions, radio propagation anomalies and noise. A photo showing the potted LAMPS tag II is shown in Figure 1. The LAMPS type II tag for example can be easily attached to any cap-lamp battery cables as is shown in Figure 2.

The intrinsic safety assessment of the LAMPS type II tag has been completed by the Safety in Mines Testing and Research Station (SIMTARS) (see http://www.nrm.qld.gov.au/simtars/). The certificate of conformity is available for inspection at http://www.mining-automation.com and following the link to the LAMPS project. In 2004, the LAMPS type II tags are priced by MineCom at less than $100 each. Since the tags are predicted to last for over two years, the annual cost of increased personnel safety is less than $50 per person.

The primary application of LAMPS is to assist with personnel safety management. The system can provide last time and identified location of personnel. This information can be used to ensure that there is adequate provisioning of safety equipment underground. Some mines have up to 30 km of underground roadways. Consequently, considerable time is often spent searching for underground equipment. Since the LAMPS type II tags are standalone, they can be used to track any mobile assets such as vehicles, which is expected to have a productivity benefit.

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The intrinsically safe LAMPS type I reader

An external view of the LAMPS type I reader is shown in Figure 3. The readers have three pairs of optical fibre HFBR 1414 transmitters and HFBR 2416 receivers for interconnection with other readers or computers. A media converter has been developed for RS232 and RS485 connectivity. The output of the HFBR 1414 optical fibre transmitters is around – 12 dB, whereas the sensitivity of the HFBR 2416 optical fibre receivers is at least – 30 dB, which provides a dynamic range of at least 18 dB. Allowing about 2 dB for connector losses, the length of fibre optic cable between readers can be at least 4 km, which is consistent with the gate roads of longwall mines.

An intrinsic safety assessment and certification of the LAMPS type I reader has been completed by SIMTARS. It is recommended that readers are installed in pairs, which enables the direction of personnel travel to be inferred. Walls, pipes, conveyors and other obstructions in close proximity to readers can reduce the available tag detection range. Therefore readers should be installed over the centre of the roadway to maximise the tag range. It is desirable that multiple cables are installed from the readers to the surface in order to reduce the impact of underground failures. For example, installing two cables to the surface as shown in Figure 4b, is preferable to installing a single cable to the surface as shown in Figure 4a.

Tag-reader performance

Aboveground trials of the LAMPS type II tags were conducted between 25 and 27 October 2004, along a meandering section of roadway shown in Figure 5. A LAMPS reader was mounted on the bottom side of a metal boom at a height of 2 m above the roadway. A utility vehicle, with various numbers of tags located on the floor of the passenger side, was repeatedly driven underneath the reader at approximately fixed speeds.

The reader was oriented with the antenna side pointing downwards. The tags were oriented upright so that the antenna side is pointing upwards. The vehicle was driven under the reader, four times, for each combination of four, eight, 12 and 20 tags, and ten, 20, 30, 40, 50 and 60 km/h vehicle speeds. The observed tag detection statistics are provided in Table 1.

The tag detection rate decreases for increasing vehicle speeds and increasing numbers of tags in the vehicle. In the case of the above-ground trials, where there are no intervening obstructions between the overhead reader and the vehicle, it was observed that up to 20 LAMPS type II tags can be reliably detected when the vehicle speed is not greater than 40 km/h. There are two factors that contribute to decreased detection rates with increasing numbers of tags and vehicle speeds. First, since the detection range is fixed, the detection time window decreases with increasing vehicle speeds. Second, the tags transmit asynchronously, resulting in more packet collisions (ie packet errors) within increasing numbers of tags.
It was observed that the aboveground detection range was around 100 m from the reader. In some underground mine trials at the Enterprise Mine in Mt Isa, detection ranges of around 50 m were observed. Therefore, it is expected that up to 20 LAMPS type tags could probably be detected at 20 km/h underground, provided that similar reader/tag orientations and geometries prevail.

**REAL-TIME RISK MANAGEMENT**

**Drivers**

Over the past 20 years, there have been an array of catastrophic incidents resulting in tragic losses of life and millions of dollars lost in capital items and sunken costs.

Spontaneous combustion events have resulted in:
- open fire leading to the closure of the Leichhardt colliery in 1982;
- CO levels in Aberdare North’s goaf rising from 100 - 3000 ppm in 12 hours – the mine was sealed at the surface; and
- CO levels at the fan in Ulan in 1991 jumped from 0 to 3000 ppm in 60 minutes with the mine remaining off-line till March 1992.

Fires at Appin 1976, West Wallsend in 1970, Liddell State in 1971 and Avon Colliery in 1976 all had significant financial and social impacts. Gas and coal dust explosions have also left their legacies on the Australian mining psyche – Appin 1979 (14 lives), Box Flat 1972 (17 lives), Kianga 1975 (11 lives), Moura No 4 1986 (12 lives), Moura No 2 1994 (11 lives) are disasters which still serve as a constant reminder of the vigilance required in underground coal mines.

Yet, close scrutiny of the various inquiries, inquests and reports lead to the observation that in many cases, predictive data was available as precursors to these events (Figure 6). However, this data was often incomplete, only available from separate and proprietary monitoring systems, could be difficult to access, and was sometimes ambiguous and often contradictory.

Further analysis of the post-incident management of these catastrophic events indicate a number of inherent difficulties faced by those charged with ‘incident management’. Such issues include having to make decisions based on less than ideal data,
these decisions are often made without the knowledge of their ultimate consequences and sequences of events once initiated, are often irreversible.

Further, the control systems in place are often algorithmic in nature and follow a simple, straight-line logic with each action preceding another, viz:

1. at the first sign of smoke don your self rescuer,
2. gather at the crib room and wait for advice,
3. if instructed, evacuate via the primary escapeway, and
4. if unable to use the primary escapeway, evacuate via the second egress.

Such procedures, whilst elegant in their simplicity, do not however, allow for the inevitable complexities that encompass an underground mining disaster. They do not allow for experience or intuition (no-one will put on a self-rescuer at the first sign of diesel exhaust), they do not provide for people to consider options or develop ‘what-if’ scenarios before taking action (why wait, for how long, what if everyone doesn’t arrive, what if someone’s hurt, what if smoke is coming but not yet arrived) and they do not allow for the human decision-making process (will they evacuate as individuals or teams, will they risk lives to help others, will they or should they be told the ‘best’ way to go).

Today, mine monitoring and data communication systems are increasingly complex and diverse. Different proprietary systems monitor the mine atmospheres, the strata devices, machinery and equipment, conveyors, pumps, fans and other infrastructure as well as report on haulage systems and belt winders. In fact, today’s modern computer networks and SCADA systems can pour over 20 000 separate data bits into our control rooms every few seconds – most of which is fastidiously collected and recorded, then studiously ignored.

People don’t have the time or capacity to digest all the data. The Nexsys™ Real-time Risk Management System seeks to provide a solution to some of these issues.

Nexsys™ Real-time Risk Analysis System

The Nexsys™ Real-time Risk Management System is a combination of IEC Ex ia (intrinsically safe) certified hardware devices and integrated software programs.

Funded by the Japan Coal Energy Center (JCoal), CSIRO Exploration and Mining and the Australia Coal Association Research Program (ACARP) the project is in its third year of development. Field trials and installation tests are being conducted with the invaluable support of Anglo Coal’s Grasstree mine with a recent installation being trialled at the Kushiro coal mine in Japan. Plans are underway for the installation of the system at a third (yet to be selected) operating longwall mine in Australia.

The objectives of the project are to:

- deliver step-change capability in critical monitoring, data integration, decision support and personnel safety;
- development of an intrinsically safe communications backbone using broad-band, high speed ethernet protocols and fibre-optic data highways;
- provide real-time hazard analysis and risk profiling; and
- develop ‘knowledge management’ capability.

- must be in use as part of normal day-to-day activity.

Developments to date – hardware

Key to the delivery of risk management critical data to the end-user, is a robust, high speed, intrinsically safe communication backbone. It was determined early in the project that the most suitable communication system would be the one that the world at large has embraced and which appears to have unlimited potential – the ethernet.

To this end, two devices have been developed. The first is a communication protocol converter (Figure 7) that can convert the serial communications protocols used by most current-loop sensors, into the language of the ethernet – TCP/IP and UDP. For those not so technically minded, it is in effect a language translator. This first such device developed was a Modbus Serial into Modbus Internet Protocol converter (akin to an Spanish-English translator) but recent developments have seen the device further developed with the capability of converting any serial protocol into internet protocols (a universal language – English translator).
The second device is a high-speed, fully managed ethernet switch. Using the latest in fibre-optic transceiver technology, the switch can be configured in any combination of 10 Mbps and 100 Mbps transfer speeds over either multi-mode or single-mode fibre. The switch has up to four ports and can be daisy-chained together and provide true V-LAN trucking capability.

Both powered by independent IEC Exia power supplies, these two IEC Exia designed devices will provide the capability to connect any serial output monitor and/or sensor with any other ethernet enabled device onto a single, high speed communication highway that is not limited by distance, will remain active during power loss, ventilation failure or the accumulation of explosion atmospheres and provide multiply redundant pathways and routing.

A further hardware device developed as part of this project is the e-Reporting System (Figure 8). This device is a stainless steel tablet that captures handwritten information, such as deputy reports, production reports, maintenance reports, etc that are typically written underground and transferring then instantaneously across the LAN. In this way, people can be made aware of the status of the section and/or schedule the next shifts maintenance and production based on the latest information. The time differences between the writing of a report and its posting on the surface, together with the time consuming tasks of contacting people underground so that the next shift plan can be drawn and the crews briefed represents a significant cost that the e-Reporting System could help alleviate.

Developments to date – software

Integration

One of the early challenges facing the project was establishing a means whereby the critical data from the different proprietary systems could be identified, sourced and integrated into a single set for further analysis and query.

The key deliverable from ACARP Project No 12011 Mine Integration of Robust Gas Monitoring and Communication was a generic group of software 'connectors' that can be installed at any mine and configured to connect a central data-base to the range of other data monitoring and collection systems (eg CITECT, Macro-view, Access, any OPC compliant systems and SQL database). This group of modules – referred to as the Integration Layer thereby provides for a single database to interrogate and read all of the safety critical data sets from there different source systems and separate them from the noise of 20,000 other data bits typically being monitored at today's modern mines.

Application

A software system known as the Nexsys™ Real-time Risk Management System is the cornerstone of the project.

The role of the Nexsys™ Real-time Risk Analysis System

The Nexsys™ system is a risk and hazard management tool for underground coal mines. The tool monitors real-time critical data from a collection of sources and detects potentially hazardous combinations of mine conditions. Mines will be able to define their own critical sources of data from any monitoring system they have in use, such as ventilation, strata, and pre or post-gas drainage systems, etc. Users can define rules for combinations of data sources. A rule inference engine continuously checks whether the prevailing mine conditions satisfy the rules and if warnings or alarms are warranted. In the event that alarms are raised, appropriate diagnostic guidance and trigger action response plans are then communicated to the appropriate personnel.

The system resides in the control room but, through its client-server architecture, can be accessed by any personnel connected to the LAN either on-site or remotely. Site senior executives and mine managers are able to view and analyse hazard profile data in a way that provides a big picture overview of the current risk status of their operations, to analyse current circumstances and to act upon any risks via the integrated decision support capability.

Statutory inspections are a critical necessity to a mine’s operation. The system includes capabilities to record and report any current and/or potential hazards in real-time to a mine-wide reporting system. Coupled with multilevel mine plans showing (among other things) the location of safety equipment together with interfaces to personnel/vehicle location monitoring, the system should reduce the level of unknowns when emergency incidents occur.

A quick tour of Nexsys™

Nexsys™ automatically uploads the latest mine workings plan. This feature is not currently available within any real-time SCADA system. This feature allows the user to: view and navigate through a three dimensional view of the mine plan; zoom in/out to/from particular locations; fly through the mine manually or via a sequence of waypoints; specify a district; turn on/off different layers of the mine plan, and manage the decision support capabilities.

FIG 8 - e-Reporting System.
Fig 9 - Viewing mine plans.

Fig 10 - Viewing sensor information.
REAL-TIME RISK ANALYSIS AND HAZARD MANAGEMENT

FIG 11 - Viewing personnel location.

FIG 12 - Viewing decision support rules.
The main graphical user interface system is shown in Figure 9. The mine plan can be seen in the main window display. All mine plans are automatically updated from a surveyor drawing file (ie a dxf file). Users can select to view various levels of the mine plan via the use of the check boxes shown on the top right-hand-side of the figure. The navigation controls can be seen along the bottom left-hand-side of the figure.

Any number of SCADA systems can be connected to the Nexsys™ system and selected sensor information displayed in one composite view. An example screenshot of sensor information is shown in Figure 10.

Nexsys™ can connect to location monitoring systems and display the last known location of personnel, vehicles and equipment. The location information is displayed as icons on the 3D mine plan. As is the case with sensor icons, users can click on location icons to access corresponding textual and image information as shown in Figure 11.

The decision support capability provides users with current status information and notification when rule criteria have been met. A combination of equipment, gas, ventilation, geotechnical and location information can be used to within rule definitions. An example rule definition is shown in Figure 12.

CONCLUSION

This paper has reported the key outcomes from two recently completed ACARP projects.

The LAMPS project has produced intrinsically safe tags and readers. This is the first location system that can be installed in hazardous regions of underground coal mines. The system reports the last known location of personnel. In 2004, tags are priced at less than $100 each. Since the tags are predicted to last for over two years, the annual cost of increased personnel safety is less than $50 per person.

Knowledge of the last known location of personnel and equipment is only a part solution to the problem of improving the management of safety. The Nexsys™ developments provide a step change in risk and hazard management. In particular, the system performs the functions as follows:

1. it integrates multiple disparate systems, including LAMPS and SCADA systems such as CITECT, within one common application;
2. the latest multi-layer mine plan information is automatically uploaded and accessible in the control room; and
3. provides a decision support capability in which a rule inference engine filters incoming mine data, checks whether hazardous conditions exist and optionally provides notification of warning advice including trigger action response plans.

However, irrespective of the introduction of new technologies, the responsibility for managing risks remains on the shoulders of our proactive and vigilant workforce.
Research Needs in Regard to Design, Performance Criteria, Construction, Maintenance Assessment and Repair of Coal Mine Seals

R Gallagher1

ABSTRACT

Legislation introduced for Queensland and New South Wales Coal mines provides different levels of prescription regarding specification of mine seals – generally in relation to capacity to withstand overpressure.

In Queensland, the Coal Mining Safety and Health Regulation (2001), Section 341 (d) places a further onus on the statutory ventilation officer to ‘ensure all ventilation control devices at the mine are properly constructed and maintained’ and that the ventilation officer ‘must ensure a ventilation control device mentioned in the regulation ... and installed at the mine meets the design criteria stated’ for the ‘type of device’.

There is limited or no prescription in regard to:

• standards and methods for design;
• standards and methods for testing of seals in the ‘lab’ and relating results to field conditions (albeit limited recognised standard test facilities exist);
• standards and methods for testing of seals in the field;
• requirement to consider specific product types in light of the particular application and specific locational environment;
• site selection for the seal;
• consideration of the operational environment of the seal;
• consideration of potential water head applied to the seal;
• control and state of the ground surrounding the seal;
• testing/acceptance criteria for a given seal, identification of defects in installation eg filling voids, etc (other than generic product tests in regard to overpressure/leakage, which may or may not bear relevance to the specific coal mine application, environment and service duty);
• seal leakage limits (although NSW uses the term ‘airtight’);
• requirement for and systems to maintain the seal, whether at design rating or otherwise;
• guidance in regard to criteria for decisions to repair or identify the need to replace seals;
• acceptable and effective methods to complete repair of seals; and
• need for methods to assess effectiveness of repair in regard to both leakage and overpressure rating.

Based on the experience of the author, it appears that risk management and life cycle approach to seals has not been adopted to the same extent as for other aspects of operations. Further, because of the number of disciplines and personnel involved that may influence various factors affecting seal integrity, the opportunity for oversight or unclear allocation of responsibility is considerable.

While suppliers can provide explosion or design rated seals, this should only be a starting point for application of the product in a coal mine. Application is often considered by mine planning staff (taking into consideration mine environment parameters such as water, control of gas, spontaneous combustion risk, etc) in conjunction with the colliery ventilation officer, and then construction completed by contractors under supervision of operational staff. After construction, seals often are managed by operational personnel with input from the ventilation officer. Input of the geotechnical engineer into pillar design, roadway opening size, ground support specification and most stable seal location is also required. The need for a customised design approach for each seal site is proposed in order to take into account the many and variable factors that may influence a site so that improved seal performance reliability and predictability can be developed.

A program of quantitative as well as qualitative monitoring of performance and triggers for rectification or maintenance action is required, and would provide support to the aforementioned proposal. Many mines rely on visual/audible inspection and periodic bag sampling as the primary means of assessment. Other significant factors such as seal material properties, rib degradation, convergence, floor heave, effects of water on both structural integrity of the seal as well as the air tightness of the seal do not generally receive the same level of attention.

Based on the above observations, the author has compiled a reference checklist in regard to the above matters, including aspects of and approaches to mine and pillar design, geotechnical modelling and data collection, civil engineering design, site evaluation and practical options available.

It is apparent that while some research has in isolation examined issues such as overpressure resistance, leakage performance, seal materials, rib sealing, effects of longwall mining and assessment of seal construction and integrity, further research may be required to deliver answers to many of the issues identified above to assist the industry and service providers develop and improve standards.

INTRODUCTION

Based on considerable operational experience and more recent completion of work as a consultant, it has observed that the level of effort, understanding and sophistication in design, installation and management of goaf seals is largely limited in focus to development of written management plans and procedures, and is often reactionary to development of alarm level conditions. Significant benefit may be gained through full life cycle consideration of seals on a panel by panel basis, and on both the planning/evaluation of likely service duty and customised selection of the most appropriate type of seal on location by location (ie individual) basis.

There exists much of the data and information required for such analysis for most mine sites (particularly for mine environment/conditions and mine design issues). For improved risk management, there is a need to develop a comprehensive data set on each type of seal available in the market. Such seal related data should include information on associated material properties (both at component and fully constructed scales), limitations or risks in use and engineering design calculations and supporting test certificates. Some, but not all of the relevant geotechnical data (such as loading and convergence experienced by seals, rib softening, geological and geotechnical immediate roof and floor strata unit models, etc) required for analysis of seal integrity may not yet be routinely collected at all sites.

Issues are discussed in relation to the elements which may be considered in improving the system of seal life cycle management. Additionally, factors expected to further drive the need for improvement and related research are outlined.

LEGISLATIVE ENVIRONMENT – QUEENSLAND AND NEW SOUTH WALES

The relevant legislation in regard to seals in underground coal mines varies between the Queensland and New South Wales. In both states, the coal mining legislation holds specific requirements with the onus for compliance primarily resting with the operators.

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In Queensland, the Coal Mining Safety and Health Act, 2001 places requirements on parties other than the operator through Section 43, ‘Obligations of contractors’, Section 44, ‘Obligations of designers, manufacturers, importers and suppliers of plant, etc for use at coal mines’, and Section 45, ‘Obligations of erectors and installers of plant’. In New South Wales, the Occupational Health and Safety Act 2000 holds precedence over subordinate and related coal mining legislation. Section 11 of the Occupational Health and Safety Act 2000, (‘Duties of designers, manufacturers and suppliers of plant and substances for use at work’) provides for similar obligations as in Queensland.

Section 325 of the Coal Mines Safety and Health Regulation, 2001, ‘Types of seals for particular circumstances and parts of mines’, states:

1. The underground mine manager must ensure a seal installed other than at the surface, at the mine is of a following type:

   a. if the level of naturally occurring flammable gas at the mine is insufficient to reach the lower explosive limit for the gas under any circumstance – type B;

   b. if persons remain underground when an explosive atmosphere exists and there is the possibility of spontaneous combustion or incendiary spark or other ignition source – type D;

   c. for an underground mine, or part of an underground mine, not mentioned in paragraph (a) or (b) – type C.

2. The underground mine manager must ensure a type E seal is used for sealing the entrance to the mine mentioned in section 156(2)(b).

Section 350(1) of the Coal Mines Safety and Health Regulation, 2001, ‘Installing ventilation control devices’, states:

1. The ventilation officer must ensure a ventilation control device mentioned in schedule four, column one, and installed at the mine meets the design criteria in schedule four, column two, opposite the type of device.

Table 1 sets out Schedule 4 of the regulation. Significant obligation is assigned to the ventilation officer in meeting compliance.

The only specific legislative requirement for seals in NSW underground coal mines is Section 99(3) of the Coal Mines (Underground) Regulation 1999, which states that:

A stopping constructed for the purpose of sealing off a part of a mine must be substantial in structure, airtight and designed to resist damage in the event of an explosion. Provision to allow sampling of the atmosphere in the sealed off area must be made.

It is apparent that the legislation is far less prescriptive and far more open to interpretation.

### SEAL LIFE CYCLE

The generic life cycle of a seal may be summarised as follows:

- consideration of impacts of mining environment and mine design on seal application, including geology, geotechnical, hydrogeological and goaf/pillar loading impacts;
- specification of operating environment, overpressure and permissible seal leakage;
- consideration of alternative materials and construction methods that may meet requirements;
- assessment of potential failure mechanism of seal in given location and likely repair and/or replacement strategy;
- selection of contractor to install preferred seal type, specific site selection, construction;
- inspection/approval of construction as in accordance with design, documentation/records of construction;
- installation of monitoring instrumentation and commencement of inspection and monitoring regime and reporting;
- completion of service in main-gate of panel providing first side abutment loading;
- service in tailgate of panel providing second front abutment loading;
- repair and/or replacement as indicated by monitoring/inspection;
- service in goaf of panel providing second side abutment loading/double goaf loading until failure or seal well inside deep goaf; and
- review of seal performance, identification of design, construction, monitoring and/or repair improvement.

### TABLE 1

Ventilation control devices and design criteria.

<table>
<thead>
<tr>
<th>Column 1</th>
<th>Column 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ventilation control device</td>
<td>Design criteria</td>
</tr>
<tr>
<td>Brattice line or temporary stopping</td>
<td>Antistatic and fire resistant</td>
</tr>
<tr>
<td>Mine entry arlock</td>
<td>Capable of withstandig an overpressure of 70 kPa while it is open</td>
</tr>
<tr>
<td>Separation stopping for a primary escapeway</td>
<td>Antistatic, fire resistant and of substantial construction providing for minimal leakage</td>
</tr>
<tr>
<td>Stopping, overcast or regulator installed as part of the main ventilation system for a panel</td>
<td>Capable of withstanding an overpressure of 14 kPa during the life of the panel</td>
</tr>
<tr>
<td>Type B seal</td>
<td>Capable of withstanding an overpressure of 35 kPa</td>
</tr>
<tr>
<td>Type C seal</td>
<td>Capable of withstanding an overpressure of 140 kPa</td>
</tr>
<tr>
<td>Type D seal</td>
<td>Capable of withstanding an overpressure of 345 kPa</td>
</tr>
<tr>
<td>Type E seal</td>
<td>Capable of withstanding an overpressure of 70 kPa</td>
</tr>
<tr>
<td>Ventilation ducting</td>
<td>Antistatic and fire resistant</td>
</tr>
</tbody>
</table>

R GALLAGHER
The level of detail and depth of investigation and analysis of these steps generally varies widely between sites in different aspects.

From observations in the working environment, seal performance between successive cut-throughs may vary considerably despite almost identical use of materials, means of construction and dimension. Often a contract is let for one type of seal for each longwall panel cut-through location, however this may not always be an appropriate approach.

An alternative approach might consider the mine environment and design, seal design, specification, seal type, site selection, construction and monitoring where construction may be varied on a seal by seal basis (ie customised) if performance is to improve and become more reliable. This would require consideration of a number of interacting factors that relate to the mining environment and the mine design.

**MINING ENVIRONMENT**

**Depth of cover**

Depth of cover will influence conditions of potential for rib spall, roof-floor convergence and/or floor heave. As stress increases with depth, effects of convergence and rib expansion and spall may become more pronounced.

**Water**

Wet conditions may impact floor conditions, the ability to prepare the site for seal installation, and the specifications of the seal (eg as a bulkhead and/or inclusion of a water trap).

**Seam thickness**

Seam thickness will influence the height of the seal, the risk of buckling and possibly method of construction.

**Seam floor structure contours**

Seam floor structure contours will indicate the likely grades and potential additional precaution required in provision of seal roof/floor frictional contact such as additional bolts, etc. Floor structure will also indicate potential areas where steeper gradients may require significant modification of design of the seal or supplementary measures to ensure stability and performance. Steep grades also raise the possibility of shear failure of pillars.

**Seam gas**

The gas content and composition will influence the risk and level of control that seal performance will be required to service.

**Seam propensity to spontaneous combustion**

Seam propensity to spontaneous combustion will influence the emphasis on both the explosion resistance and air tightness required of the seal and surrounding ribs, and possibly modify approaches to rib support and/or grouting.

**Geological structure**

Detailed exploration and underground mapping during development operations will identify potential areas where abnormal ground behaviour or other conditions may eventuate on longwall extraction that can impact on seal performance.

**Stratigraphy/immediate roof and floor**

Detailed modelling of geological and geotechnical roof and floor units will assist in anticipation of problem areas such as seam splitting or the presence of rider seams, as well as identify potential zones for poorer roof or floor conditions and potential horizons of shear failure in future longwall mining pass bys. Such information may significantly impact the type of seal selected for a site and influence the ground support strategies applied in these areas.

The majority of the required information is collected in the course of typical exploration, geological modelling and resource assessment processes for underground coal mine evaluation. It appears that there is limited further information regarding the resource that may be gathered and usefully assessed in regard to design, selection and installation of seals.

**MINE DESIGN**

**Pillar and roadway size**

The level of conservatism in pillar design (for a given roadway size, pillar size, depth and geomechanical properties) will significantly impact on assessment of required primary and secondary ground support, and in turn on supported roadway deformation and integrity. A number of pillar design methods will typically be applied to develop confidence in pillar stability. Ideally, this is verified by monitoring strata movements. Numerical modelling can provide insights not only to anticipated roadway deformation and support requirements but also as to aspects that may influence duty conditions for seals such as convergence, likely roadway failure modes and failure locations, etc. Experience in numerical modelling of stress conditions for successive longwall panels with the inclusion of criteria of limiting stress values aids selection of the location of seals as a factor in pillar size assessment. The limiting values may be derived through actual seal performance assessment and completion of modelling in the same package to derive the threshold. Indication of preferred location of the seal in the cut-through should also be possible.

**Ground support and timing**

Typically seals are installed after secondary support has already been installed. On initial development mining, the mining method (eg in place versus place change mining) and timing of primary support installation may impact levels of immediate roof delamination which can significantly impact the secondary support requirements and performance as well as ultimate roof behaviour. Data such as convergence (eg tell tale readings or roof extensometry) is typically taken at intersections and is relevant in considering variations in supported ground behaviour at different cut-throughs. Ongoing review and specification of ground support requirements, particularly in poorer ground can significantly impact on subsequent seal performance.

**Rib control**

Mines typically collect rib extensometry data to verify design and optimise rib support. Rib control in the cut-through locations where seals are to be installed can be critical, particularly at depth. Where softening depth is difficult or uneconomic to control, there may be little option other than to grout or inject the ribs. From experience, typically a distance of 10 - 15 m either side of the centreline of the seal (approximately twice depth of softening) may be required to circumvent leakage. In extreme cases, reinforcement, shuttering and pouring of artificial rib followed by pressure grouting through the artificial rib may be required. As an example, measured rib softening in excess of 7 m has been found. Following change to rib strapping rather than spot bolting (providing greater confinement), rib softening was reduced to 0.5 m. Such improvements can have a large impact on rib leakage around seals and overall integrity of the seal/rib contact and cut-through in general.
Mining height
Where mining height is less than seam height, an indication of the amount of roof and/or floor coal that will need to be dug out or cut down during development can be made on a localised scale and also included as part tender specification in provision of services for installation of seals by contractors.

Seam dip and cross-grade
Cross grade will determine whether water will tend to pool in roadways, against the inspected side of seals (with the seal on the downdip side of operations) or against the goaf side of the seal (with the seal on the uphill side of operations).

Panel grade
A long section profile with exaggerated vertical scale should be generated to identify those locations planned for seals which will act as collection points for water (swillies) and to enable estimation of maximum head of water that the seal may be subjected to before water will flow to a lower elevation.

Gas drainage
Where gas drainage is applied, there will be associated dewatering and potential coal shrinkage which may impact local strata conditions. Care should be taken to ensure that both surface and in-seam boreholes are sealed so that they do not represent potential leakage paths. Particularly where holes pass within 10 - 15 m of seal locations, rib grout injections should be used to prevent leakage modelling following an overpressure event as a number of other factors including ground conditions and seal/rib/roof/floor unit is a potentially useful extension of the application. Modelling however appears an unlikely predictive tool for leakage and loading of the stopping and that anisotropic loading may lead to premature failure; lateral movement up to ~30 mm for this type of stopping is possible without apparent substantial damage; that the stoppings could resist vertical loads of 2700 - 3000 kN (which was almost three times the maximum loadable to be carried by an individual lightweight block); that insertion of a phenolic foam yield layer in the stopping allowed for initial convergence but also allowed some block rotation that reduced the capacity of the stopping to cope with convergence; that wedging of rows of lightweight blocks (ie providing lateral confinement/loading) plays a substantial role in the ability of the stopping to resist lateral and vertical loads.

The collection of this level of detail from data from an industry survey would be a positive step in provision of data upon which an empirical or mathematical modelling tool could be developed. A survey by Oyler et al regarding the perception of validity of this modelling by mining companies indicated that nine out of 14 are supportive. Testing authority certification: what does it really mean?
There are differences in results derived from different testing authorities based on the physical geometry/dimensions of the authority, as well ad the resultant explosion. Oberholzer and Lyne (2002) describes this aspect in some detail. All facilities currently utilise physical explosion tests, although Sapko et al (2003) provides details of a hydrostatic test method which has recently been trialled. It is worthwhile briefly revisiting the origins of overpressure specification of seals and assumptions made in conjunction with the values derived.

The National Institute for Occupational Health and Safety (NIOHS, 2001) report a brief summary of goaf gas explosions which occurred in US coal mines, resulting in destruction of goaf seals. Lightning strikes were identified as the likely energy source, with transmission to the goaf area postulated as steel cased boreholes. Stopping material fragments were strength tested to provide a guide as to the seal strength that may be required to avoid destruction. It was concluded that:
- seal strength of minimum 20 psi for mines without explosive mixtures of flammable gas and 50 psi for mines with explosive mixtures of flammable gas are appropriate;
- pressure balancing of the goaf to reduce oxygen ingress and size/opportunity of accumulations of flammable gas in the explosive range is required;
- deep steel casing connecting the surface to the goaf, particularly in the vicinity of locations where explosive mixtures of flammable gas may accumulate should not be used; and
- a high standard of stonedusting on the inside and outside of location of the seals prior to installation is required.
Relating each of these recommendations in turn to observed industry practice:

- Queensland has adopted specific overpressure standards, whereas New South Wales has adopted a less prescriptive standard;
- in regard to the necessity of pressure balancing, Pearson et al (2000) received a very mixed response to this an essential requirement in a survey of Australian mines;
- in regard to steel cased boreholes in proximity to explosive gas risk zones, many Australian mines use goaf drainage and/or have fully cased surface boreholes adjacent to these areas; and
- observation suggests that industry typically does not follow this recommendation religiously, and that in many mines, rib spall may quickly negate the effect of once off stonedusting of ribs.

It appears that a majority of industry focus has been on the overpressure and leakage testing of seals based on the above recommendations; however, the recommendations are not generally implemented by the Australian industry. The industry might consider questioning the value of certification particularly in relation to the following:

- Various modifications (eg sampling pipes, bleed pipes, water traps, etc) being added to the seal certified by laboratory testing without additional test work. Note that doors are reportedly not considered by most suppliers as impacting integrity (Pearson et al, 2000), however no test results were made available to that survey.
- Procedures are not in place to ensure that seals are adequately constructed and maintained. The act of construction is not generally backed up with quantitative means of testing the seal or other means for confirming that a defect is not present. Ongoing compliance seems to be a significant issue that is largely without guidelines, somewhat overlooked and without sound controls.

Stephan (1990a and 1990b) reported in the United States Mines Safety and Health Administration – Ventilation Division, seal test work and the original mine explosion assessment and research work regarding the recognised standard for assessing damage to seals and identification of suitable means for repair and methods for recertification following repair.

Stephan (2004) indicated that:

> Arbitrary decisions are made based on the visual observations of the seal’s condition. Seals are to be maintained in a condition where they remain able to withstand 20 psi overpressure. The pass/fail nature of seal construction is based on the seal’s ability to resist air leakage after impacted by such an overpressure. Small cracks may be okay but loose or missing blocks would cause the seal to be considered out of compliance. There is no damage assessment standard for seals. Suitable means for repair are not specifically identified.

In response to the enquired as to whether Mr Stephan was aware of any work completed either as research or in the field in regard to relationships between load or convergence monitoring, damage to seals and assessment of ongoing explosion rating and or leakage, Stephan responded that:

> A ‘recertification’ of repaired seals is not in place. If excessive leakage occurs, for any reason, repair or replacement of the seal will become necessary. There are no guidelines for how to accomplish this task and no specific definitions as to when repair or replacement is necessary.

**Site selection**

Ideally, site selection within the cut-through is aided by analysis of data including stratigraphy (preferably derived from roof and floor core) and geological mapping (at detailed level). Floor grade, cross grade and water conditions require consideration and may influence the type of seal built at low points. Suitability of roof, floor and rib conditions and preferably a point of reduced width is useful. The flatness of floor and roof surfaces will also impact the ease of constructing and sealing the seal.

**Ground and seal material properties**

Coal mine strata are not always ‘stiff’ (eg coal, laminites and clays versus sandstones or conglomerates). Usually, the floor in a coal mine is more stiff than the immediate roof.

In civil engineering design, combined stiff and yielding systems rarely provide an appropriate solution for a given support problem. This leads seal material property specification into difficult choices, as most solutions are actually a combination of stiff and yielding systems. The yielding elements of the systems are usually limited in capacity and provided to allow absorption of a certain degree ground movement. The ultimate ideal balance is dictated by the service duty requirements of the individual mine.

To illustrate some of the advantages and disadvantages of seal material combinations, a brief summary description of US tested seal types follows.

**TYPES OF SEALS TESTED IN THE LITERATURE**


- 6 × 8 × 16’ solid block;
- keyed to roof and floor with timber, wedged to the roof with timber to provide confinement; and
- use of central pilaster for span protection – critical in seal strength.


Requires framing construction (typically props, battens, ply).

- Strength is subject to mixing, curing time and conditions during curing. Density/strength control is critical, and samples of the mixed foam should be taken and tested to verify strength.
- Can mix and pump some distance to use one location setup.
- Effects of water build up (especially acidic water) behind this type of seal should be avoided.
- Can increase friction between rib and cement plug by placing protruding rib bolts.
- Formwork and foam retention materials need to be removed to enable inspection of seal.

**Lightweight (Omega 384 – glass fibre reinforced) and/or hollow block seals – Stephan (1990), Weiss et al (1993)**

- Blocks are impervious to water and air leakage,
- cure time if bonding material is applied to blocks, and
- require hitching similar to that used for solid blocks.
- Application in US in deeper operations where experience roof, floor or rib convergence (concrete block seals typically fail due to stiffness);
- typically only wedged into place row by row – require hitching;
- thick stonedust layer used to assist in rib sealing;
- stacked in direction of CT;
- sealant applied on goaf side of seal; and
- require additional retention to prevent en masse movement in event of overpressure.

- Composite system between stiff outer walls, moderately yielding aggregate fill and fairly flexible polyurethane foam binder;
- only structural element of the arrangement appear to be the block walls – once destroyed only the friction of the core against the roof/floor/rib will prevent en masse movement;
- mixing in 1:1 ratio required of the polyurethane components needs to be accurate to meet density;
- it is critical to obtain correct aggregate/foam ratios – dry bagged/sized aggregate is used;
- surfaces need to be free of debris and duct between pours;
- foam may cause bulging of block walls during filling;
- need to wait for set between pouring subsequent courses;
- moisture/humidity is an issue in regard to bonding between polyurethane layers and/or roof/rib/floor surfaces;
- polyurethane is a fire hazard – more emphasis on fire resistance/requirement for external coating; and
- core thickness (related to seal height) is a key design issue.

- Density control an issue,
- ensuring fill to mine roof can be difficult,
- strength subject to curing time and conditions during curing,
- cold joints effectively a defect at higher overpressures,
- sensitive to method of poor, and
- woven steel reinforcement can provide significant integrity improvement.

- As per cementicious foam seals but with additional strength/confinegment provided by the walls; and
- increased cost of seal.

- Meshblocks are secured to the ground through perimeter bolting with protrusions into the middle of the block;
- the meshblocks reinforce the shotcrete; and
- cost is reduced in comparison to the previous seal type.

Significant experience and data gaps exist in regard to publicly available industry experience databases regarding conditions to which seals are exposed, in particular regarding load, convergence and rib expansion/spall. Some of this data may be available from geotechnical design verification/confirmation. There appears to be a paucity of good quality data regarding seal performance, as well as definition of tolerance limits for aspects of convergence, buckling and material properties of complete seals. The variations which exist in seal design (including crush blocks or timber, location of various pipes, doors and water traps) makes performance comparison difficult, and it also appears significant attention needs to be paid to geological variation.

GROUND PREPARATION
Key issues include aspects related to achieving appropriate keying in (including depth required) and/or setting of additional support as required for frictional resistance. Removal of loose floor and rib debris, as well as cutting into coal/stone to refusal (preferably removing all roof/floor coal) is critical for overpressure resistance and leakage reduction.

A formalised system of permitting of construction sites as described by Humphries (1999) is sensible, provides hardcopy records, adds control and can include a checklist for guidance. The permit also provides guidance in relation to seal specification, installation and inspection/approval following construction.

CONSTRUCTION METHOD
Construction method is normally in accordance with the supplier’s procedure, which is typically based on engineering calculations, consideration of the material properties of the seal components and risk management approaches.

RECOVERY MEASURES
Consideration of recovery measures in the event that seal leakage and/or failure occurs in a given location requires consideration at planning stage and not after construction commences. Ideally, a strategy for each individual cut-through will be developed and when required, preparatory work completed as a part of seal construction. Examples of such works may include installation of rib reinforcement, excavation of keying in channels, construction of a containment wall on the goaf side of the seal so that the void between can be filled at a later date to create a plug seal.

Adequate space should be left for construction of another seal should the need arise.

In order to assist in dealing with leakage repair and/or development of a heating, controlled leakage measures (eg inclusion of pipe with a valve) may be used. This approach will allow for preferential leakage through the pipe, avoiding fractured coal zones and allowing completion of rib sealing.

CONSTRUCTION
Materials ease of use, logistics and handling for use underground and time for construction will be important. Control of material properties can be a critical factor, particularly where mixing grouts with water, two part resins, or needing control of placed density to ensure rating or integrity.

Particular consideration is required to be given to seals with materials that have a curing time prior to achieving full strength and/or rating. Control of conditions/environment in the seal installation location may be required (eg temperature, water, etc).

Identification of possible forms of construction defects (strength, voids, anchors, etc) needs to be completed prior to award and should be considered in the checklist and methods for reviewing construction activities and completion.
Again, use of a permit system to provide documentation, guidance in regard to critical factors influencing construction and resultant seal compliance with test certification and acceptance criteria/inspection records is essential. Humphries (1999) also indicates that a calculation of likely load on the seal from longwall abutment loading is also required. While of interest, unless the effective strength of a seal is known, as well as the impact of other factors, the factor of safety (and failure risk) for a seal cannot be estimated.

The mine typically remains responsible for evaluating, specifying and installing additional ground support required to protect the seal once installed. In particular, secondary support requirements will need to consider effects of longwall mining and opening stability, protection of the seal against convergence, protection of the ribs in vicinity of the seal and protection of mine personnel from seal failure/toppling should such an event occur.

Where poor rib conditions develop, additional increment/s of rib sealing may be required through grouting, application of polyurethane foam, or rib shuttering and forming with subsequent drilling and pressure grouting.

There may be an issue with timing of damage to the ground from development to first longwall abutment loading to side abutment loading to goaf reconsolidation to second abutment loading, requiring successive ground support review and response.

**OPERATIONAL**

**Stonedusting**

As noted previously, an important element of the US approach to limiting risk of overpressure from goaf explosion propagation into the gate-road via the seals is the application of incombustible dust.

**Control of goaf gas composition and proximity of explosive mixture to seal**

Control of goaf gas composition and proximity of explosive mixture to seal is dictated by a combination of the mine ventilation system, goaf gas drainage system and standards of seals.

**Rate of seal completion**

The ability to complete construction rapidly to match longwall retreat rate is important to avoid oxygen ingress to the goaf. It is noted however that construction of seals in the maingate up to hundreds of metres in arrear of the face will experience ongoing loading as the goaf reconsolidates. This behaviour has been clearly demonstrated by microseismic monitoring (Hatherly *et al.*, 2003) at a number of sites and explains why damage can continue to occur to seals until the longwall face passes in excess of 500 - 600 m outbye.

**Impacts of successive goaf loading**

Impacts of successive goaf loading include response of installed secondary support and in particular the lateral movement towards and then away from the goaf as the second face passes the location of the seal.

**MONITORING**

**Seals**

Monitoring procedures include visual (dependent on type of seal), audible (leakage), air flow (ventilation reading, smoke tube), gas sampling – general body in the seal cut-through and from behind the seal, seal buckling/movement (from displacement of the face of the seal), load cell monitoring, convergence monitoring and rib softening monitoring.

**Ribs, roof and floor**

Visual indicators include spall, convergence/heave or water flow or bubbling from the floor. Open cracks may be observed, but deeper fracturing may be difficult to assess. Use of devices such as shear strips, etc may provide indicators of the progress and extend of rib damage. Convergence monitoring is also applicable. The mode of strata and seal failure needs to be carefully observed.

**Inspection regime**

Successful inspection regimes will include regular and appropriate frequency, and preferably use of the same personnel to complete inspections. Inspection without a specific checklist of matters to examine and record status will be far less effective than a well thought out and designed record sheet based one. Collection of goaf gas bag samples from behind seals is often included within the weekly seals inspection scope.

**Ventilation, gas monitoring and spontaneous combustion**

Aspects of routine monitoring of the mine ventilation and gas control systems including mine fan pressure and quantity, panel return pressure/quantity, panel return gas levels and various gas ratios and mass flow rates will all be useful in identifying changes in seal integrity. The reliability of gas monitoring systems also needs to be checked regularly.

An example of the effectiveness of a simple inspection system in a mine prone to spontaneous combustions and appropriate response is illustrated by Nicholls (2004) in the description of a minor heating which developed due to damage to a seal in the immediate goaf behind the longwall face.

**MAINTENANCE**

Consideration needs to be given to the type of maintenance that may be required and the materials and skills required to complete it.

There are a number of components to any seal, including: the seal wall (both overpressure rating and air tightness), devices and/or gauges fitted to the seal and pipes (including pressure gauges, level indicators or gas monitoring tubes/sensors), ground support around the seal, water traps, sampling tubes, inertisation pipes, the ribs, the roof, the floor, travel ways to/from the seal, pumping in access roadways and ventilation of the seal.

Many mines treat seal maintenance as an exercise in patching up cracks or recoating the external surface of the seal as a majority of the other matters form routine operational tasks.

The primary research need, as indicated earlier is in regard to the effectiveness of maintenance/repair and whether seal overpressure rating is retained. In this respect, there remains no clear guidance as to when a seal should be replaced.

In ACARP Report C10014, Oberholzer (2002) considers in situ test methods for ventilation structures. Initially consideration was focused on non-destructive tests, however it was broadly concluded that there was limited scope, and that destructive in situ testing (with portable test equipment) was preferred. Trevits *et al.* (2002) trialled application of ground penetrating radar (GPR) and Schmidt Hammer Tests (both non-destructive approaches) on cementicious seals. Some success was evident with both methods, and GPR in particular appeared to show promise. However, as Oberholzer concluded, the likely cost of equipment for approved use in assessment of in situ goaf seals is likely not prospective and the preferred approach is destructive testing in lower cost test galleries.
Significant further research is required to deliver means for assessment of seals in situ, for effective repair (outside of current practice) and means to be able to reassess seals as acceptable following repair. Further, means for assessing required replacement of a seal based on objective criteria is required.

INTEGRATION WITH OTHER MANAGEMENT PLANS

Consideration needs to be given to the integration with management plans that cover spontaneous combustion, gas monitoring, ventilation, gas drainage, strata management, longwall operations, mine inspection, extraction panel sealing and emergency response in order to streamline operational control and improve response and risk management. A particular example is means of provision of distribution of Tomlinson boiler gas for goaf inertisation. While the conversion and use of existing boreholes and pipelines is often proposed, the approach will not often be appropriate as the switch over/preparation time for unplanned incidents will often be so long as to allow the incident to escalate.

ISSUES FOR THE FUTURE

Australian underground coal mines are gradually becoming deeper and gassier. This requires greater care and emphasis in pillar, roadway and support design in combination with considerations for mine seals. At increased depths, floor heave and rib expansion/spall effects will also continue to increase in severity and increase risk.

Further, the industry appears to be embarking on the path of operating multi-seam longwall workings. Where the interburden between goaves is relatively thin (30 - 50 m), sealing the overlying goaf from the caved area of the undermining seam may be problematic, and new solutions will need to be found.

The push for productivity and cost reduction is driving a trend for longer and wider longwall panels, higher ventilation pressure differentials and a squeeze on both site based professional staff (in terms of both numbers and adequate time to appropriately complete all assigned tasks) as well as cost of seals and physical inspections.

A system design approach is essential for the future so that important issues are adequately considered at an early stage of planning to reduce reactionary problem solving.

REFERENCES


Development of Innovative Goaf Inertisation Practices to Improve Coal Mine Safety

T X Ren1, R Balusu2 and P Humphries1

ABSTRACT
In combination with detailed field studies and goaf gas characterisation, computational fluid dynamics (CFD) models have been used to develop optimum and effective strategies for inertisation during longwall sealing operations to achieve goaf inertisation within a few hours of panel seal-off operation. This study has combined detailed analysis of the performance of various inertisation field trials together with CFD modelling results of different inertisation operations in order to develop the optimum inertisation strategies. A number of parametric studies were conducted on the base case CFD models that had been calibrated and validated based on the information obtained from previous inertisation studies and goaf gas monitoring. These studies included changes in inert gas injection locations, inert gas flow rates, seam gradients, and different inertisation strategies to investigate their effect on goaf inertisation. Studies indicate that the strategy of inert gas injection through the MG seal was not as effective as the alternative strategy of inert gas injection at 200 m behind the face. Innovative inertisation strategies have been developed and subsequently implemented at an underground coal mine in Australia.

Further investigations have been carried out in the development and demonstration of proactive inertisation strategies with the objective to suppress the occurrence of spontaneous heatings in active longwall faces, in particular under unexpected scenarios such as during slow retreat/face stoppage due to difficult geological conditions. Initial trials at two Australian longwall panels have demonstrated the great potential of this practice to contain the onset of heatings in the goaf.

INTRODUCTION
Goaf inertisation with inert gas has been used worldwide to control active fires and spontaneous heatings in underground coal mines. In Australia, nitrogen injection was used in number of mine fire incidents, with varying degrees of success. For instance, nitrogen was pumped into Moura No 4 mine after the explosion to render the mine atmosphere inert for rescue teams to enter and control an active fire created by the explosion (Lynn, 1987). Inert gas was also used at Ulam Colliery to control a major spontaneous combustion incident (Healey, 1995).

This technique is being deployed to lower the risk of potential explosions during longwall panel sealing off periods. In Australia, inert gas from Tomlinson boilers and drained inseam gas are being used in some mines for routine inertisation operations. The specific objective of inert gas injection operations is to reduce the goaf oxygen levels below the safe limit of eight per cent (ie with a factor of safety of 1.5 on the explosive nose limit of 12 per cent) before methane concentration reaches the lower explosive limit of five per cent. The inertisation schemes usually involved injecting inert gas through maingate (MG) or tailgate (TG) seals until goaf gas sampling results show that oxygen level was below eight per cent. In many cases it was found that the goaf oxygen concentration was above 12 per cent even after two to three days of inert gas injection and in some cases an explosive atmosphere was also present in the goaf during inertisation. There was a need to optimise inertisation operations to reduce the goaf oxygen levels, thus reduce the explosion potential as quickly as possible during longwall sealing off periods.

Recently, the occurrences of spontaneous combustion in longwall goafs has led to mine abandonment or production suspension in a number of underground coal mines worldwide. Proactive goaf inertisation for open goaf in active longwalls can be used to suppress the development of potential goaf heatings and ‘save’ time for the longwalls to advance beyond dangerous zones and to sustain normal production rate. This method is particularly important for reducing the risk of spontaneous heatings in active longwall goafs during slow face movement due to geological difficulties, ie faults/roof falls, roadway collapse or other production problems. Supported by ACARP and in collaboration with Australian underground coal mines, CIISO is in the process of developing proactive inertisation strategies with the objective to reduce the risk of spontaneous heatings in active longwall faces, in particular under unexpected scenarios such as during slow retreat/fce stoppage. Initial trials of the proposed proactive inertisation techniques in two Australian underground coal mines have shown promising results.

This paper provides a brief review of the traditional inertisation practices during longwall sealing operations in Australian coal mines and the applications of CFD models to understand the gas flow mechanics and distribution patterns in longwall goafs. The paper discusses the simulation results of various goaf inertisation strategies for face seal-off operations and active longwall goafs (proactive inertisation). Optimum inertisation strategies were developed and demonstrated in field demonstration studies.

REVIEW OF TRADITIONAL INERTISATION SCHEMES
Longwall goaf inertisation has been carried out on a regular basis in some Australian mines to reduce the potential risk of explosions during the panel sealing-off period. Traditionally liquid N2 and CO2 were used in most of the fire control inertisation operations. However, it was difficult and expensive to procure large quantities of the inert gases for routine longwall sealing applications, particularly in mines located at remote places of Australia. In 1997, the Tomlinson Boiler low-flow inertisation device and a high capacity GAG 3A jet engine system were demonstrated to the Australian mining industry as new practical tools for inertising underground mine atmosphere. The successful demonstration of these devices has improved the availability of inert gases for routine mine applications.

Over the last few years, there have been over ten applications of inertisation during longwall sealing operations. Analysis of the data from some of the mines showed that the inertisation schemes implemented were not effective in preventing the formation of explosive gas mixtures near the longwall finish line for up to two days after panel sealing. In one case, the goaf atmosphere near the finish line fluctuated widely and the oxygen concentration was over the 12 per cent level a number of times over the two week period after sealing. Results from another mine showed that although the inertisation schemes employed at that mine were relatively more effective when compared with results of other cases, oxygen levels in the goaf were still above 12 per cent for up to two days after panel sealing.

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2. MAusIMM, Principal Mining Engineer, CSIRO Exploration and Mining, PO Box 883, Kenmore Qld 4069.
In a typical inertisation practice, inert gas is injected into the goaf, mostly through the MG seal immediately after sealing the panel. Recently, some mines started the practice of injecting inert gas simultaneously into both MG and TG seals or other seals depending on the oxygen levels at various locations around the goaf. The inert gas generator is normally set up at a temporary surface site above the longwall and one or two 150 mm diameter boreholes are drilled from the surface into the roadways for inert gas delivery.

In the typical case presented here, the maingate was used as an intake airway and the tailgate as return airway during longwall retreat operations. Airflow of 40 to 50 m³/s had been maintained along the face during longwall extraction. In this case, the panel orientation was such that the maingate intake was at a higher elevation compared with the tailgate roadway and the outbye tailgate corner was the point of lowest elevation. Methane gas emission in the panel was relatively low at the rate of about 300 L/s. After sealing off the panel, Boiler inert gas was injected into the goaf through the MG seal for inertisation.

In another typical case study, inert gas was injected through both MG and TG seals, immediately after sealing off the panel. Gas composition in the goaf after one day of inert gas injection is shown in Figure 4. Analysis of the results shows an increase in oxygen level to 15 per cent at 3 c/t seal, which indicates that high O₂ concentration pockets were still present in the goaf even when inert gas was injected through both MG and TG seals.

The review studies indicate that simply injecting inert gas through MG or TG seals does not achieve the objective of quick inertisation of longwall goafs. Analysis of results indicated that the effect of inert gas injection through the MG/TG seals on gas composition at inbye locations of the goaf was negligible for up to two days after sealing. There is a need for optimisation of inertisation strategies to achieve the desired objective of goaf inertisation within a few hours of sealing. This requires a detailed understanding of goaf gas flow mechanisms behind the longwall and the subsequent impact of inert gas injection on goaf gas distribution.

### CFD SIMULATIONS OF LONGWALL GOAF GAS FLOW

CFD modelling has been used in the minerals industries in a number of areas, including control of methane and spontaneous heating (Creedy and Clarke, 1992; Tauxe et al., 1993; SMRAB, 1997; Ren and Edwards, 1998; 1999), dust control...
(Aziz et al., 1993; Sullivan et al., 1993), diesel particulate emissions (Currie, 1994), mine fires and explosions (Lee, 1994), auxiliary ventilation layouts in rapid heading development (Moloney et al., 1998) and mineral processing (Fletcher et al., 1995). CFD codes have been used in Australia for development of goaf gas control (Balusu et al., 2001) and more recently goaf inertisation strategies (Balusu et al., 2002).

A commercial CFD package Fluent has been used for this study. Fluent is a finite volume CFD code that solves the Navier-Stokes equations for both incompressible and compressible flows. A key feature of this code is its user-defined function capability, or UDF, which allows the user to develop stand-alone C programs that can be dynamically linked with the solver to enhance the standard features of the code.

Gas flow migration in a longwall goaf is complicated process as many factors are involved, such as ventilation layout and intensity, gas emission rate and compositions (eg the presence of methane and carbon dioxide), face (seam) orientation and dip, gas buoyancy and goaf permeability. A range of CFD models have been developed to achieve a detailed understanding of the gas flow mechanics and distribution in longwall goafs. In addition to innovative CFD modelling, the study also involved extensive validation and calibration of initial models using data obtained from field studies and parametric studies to investigate the effect of various parameters on goaf flow patterns. Models were then used in the development of gas and spontaneous heating control strategies through simulation of the effectiveness of various designs and control techniques. The CFD modelling work generally involves a number of key stages, including:

- field studies to obtain the basic information on panel goaf geometries and other parameters;
- construction of 3D finite element model of the longwall goaf;
- setting up flow models and boundary conditions through UDFs;
- base case model simulations;
- model calibration and validation using field measured data; and
- extensive parametric studies and development of optimum strategies.

A key part of the CFD models is the incorporation of longwall goaf permeability distributions and gas emissions via a set of UDFs that are linked to the solver. Flow through goaf was handled using custom written subroutines, which were added to the ‘flow through porous media’ modules of the basic code. In these subroutines/modules, flow through the porous goaf regions was simulated by adding a momentum sink to the momentum equations. The sink had viscous part proportional to the viscosity and an inertial component proportional to the kinetic energy of the gases. A number of subroutines were written to represent different ventilation and goaf gas emissions scenarios, which were then combined with the main CFD program to carry out the simulations.
Typically the CFD models are in 3D with 500 000 cells in order to capture the behaviour of goaf gas flow in a 250 m longwall panel up to 1000 m in the direction of face advance. Longwall CFD models can be constructed according to the actual mine layouts. The mesh used in the models was ‘refined’ with higher density mesh in the areas of interest such as areas next to the face and roadways. A typical geometry and mesh used in longwall goaf gas flow models is shown in Figure 5.

**GOAF INERTISATION FOR FACE SEAL-OFF OPERATIONS**

CFD models have been used to develop optimum and effective strategies for inertisation during longwall sealing operations to achieve goaf inertisation within a few hours of sealing the panel. The study has combined detailed analysis of the performance of various inertisation field trials together with CFD modelling results of different inertisation operations in order to develop the optimum inertisation strategies.

Parametric studies were conducted on the base case CFD models that had been calibrated and validated based on the information obtained from previous inertisation studies and goaf gas monitoring. These studies included changes in inert gas injection locations, inert gas flow rates, seam gradients, and different inertisation strategies to investigate their effect on goaf inertisation. The modelling results indicated that there were no major differences in goaf gas distribution between the injection of boiler gas and nitrogen; however, different inert gas injection points resulted in entirely different goaf gas distribution.

Figure 6 shows the oxygen distribution patterns within the goaf (at the level of mining) following the injection of inert gas through maingate and the third cut-through seal respectively with steady state simulations. It can be observed that following the injection of inert gas through the maingate (MG) seal, oxygen concentration level was reduced from 21 per cent to 17 per cent only within the immediate vicinity of the maingate seal. The air and gas mixture zone with 12 per cent to 14 per cent oxygen was pushed back deep into the goaf up to 200 m – ie the explosive zone was expanded to a wider area. Whilst as the inert gas was injected via the third cut-through seal, some 200 m behind the face finish-off line, the air and gas mixture zone was pushed towards the face finish line, narrowing down of the explosive zone in the goaf.

Figure 7 shows the oxygen distribution in the goaf for inert gas injection at different locations after 24 hours of injection (time-dependent simulations). Inert gas at a rate of 0.5 m$^3$/s was injected through the MG seal and at 200 m behind the face (through 3 c/t seal) on the maingate side respectively.

Analyses of the numerical results indicate that the strategy of inert gas injection through the MG seal was not as effective as the alternative strategy of inert gas injection at 200 m behind the face (ie through 3 c/t). Analysis of the various simulation results also indicated that longwall panel geometry, goaf characteristics, gateroad conditions in the goaf, goaf gas emission rates and composition, ventilation during panel sealing off period, chock withdrawal and panel sealing sequence would also have a significant influence on goaf gas distribution and inertisation.

Based on the results of various simulations, an optimum inertisation strategy has been developed taking into consideration the positive effects of various inertisation schemes and the field site conditions. Field demonstration studies of the optimum inertisation strategy were conducted in a longwall panel of the Newlands Colliery, one of the less gassy mines in Australia (goaf gas emissions in the range of 100 L/s to 500 L/s). It should be noted that effective inertisation of a sealed goaf may take a longer time in less gassy mines. Therefore, Newlands Colliery presented one of the difficult conditions for goaf inertisation, which was ideal for field demonstration studies.

The optimum inertisation strategy developed during the course of the project for Newlands Colliery site conditions basically involved:

- inert gas injection through tailgate 4 c/t and TG seals for two days before sealing;
- inert gas flow rate at 0.5 m$^3$/h (boiler gas);
- inert gas injection through maingate 4 c/t (ie at 200 m behind the face finish line) for one day with door on chute road seal still open; and
- panel sealing and continuation of inert gas injection through maingate 4 c/t until oxygen levels in the goaf reduced below eight per cent.

Field demonstration study results show that the optimum inertisation strategy implemented at the field site was highly successful in converting the goaf environment into an inert atmosphere within a few hours of panel sealing. During these demonstration studies, results show that the goaf atmosphere was completely inert with oxygen concentration below five per cent at all locations in the goaf by the time of closing the doors on the final seals. Results also showed that oxygen levels in the goaf did not rise after stopping the inert gas injection, confirming the success of goaf inertisation.

![Figure 5 - Typical model geometry and mesh used in the longwall CFD gas flow models.](image-url)
An on-going project at CSIRO is the development and demonstration of proactive inertisation strategies with the objective to reduce the risk of spontaneous heatings in active longwall faces, in particular under unexpected scenarios such as during slow retreat/face stoppage.

CFD simulations were conducted for a range of longwall layouts and gas emission conditions based upon several Australian underground coal mines. These models were used to investigate the best inertisation strategies that could be deployed to narrow down the high oxygen level zones which are potentially liable to spontaneous combustion in the goaf. The investigation involved extensive parametric studies on inert gas compositions, injection locations, inert gas flow rates as well as the impact of goaf gas emissions, seam dips, face orientation and ventilation systems.

**PROACTIVE INERTISATION FOR ACTIVE LONGWALL FACES**

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**FIG 6 - Oxygen distribution patterns in the goaf following inert gas injection at different locations.**

**FIG 7 - Oxygen distribution in the goaf – with inert gas through MG seal and 3 c/t seal respectively (transient simulation one day after sealing).**
T X REN, R BALUSU and P HUMPHRIES

**Base model**
Boiler gas at 0.5 m/s, 30 m behind face line

**Boiler gas at 0.5 m/s, 110 m behind face line**

**Boiler gas at 0.5 m/s, 200 m behind face line**

**Boiler gas at 0.5 m/s, 500 m behind face line**

**Inert gas injection point**

**Fig 8** - CFD simulations of proactive goaf inertisation options in an active longwall.
Referring to the longwall layout in Figure 5, Figure 8 shows the goaf oxygen distribution patterns at the working level for the base model and inert gas (boiler gas) injection simulations at different cut-through seals on the maingate side. The face was ventilated with 45 m$^3$/s of air using a modified form of back return system (a cut-through open on the tailgate for return air).

The results indicate that inertisation through cut-through seals close to the face line would not be effective – most of the inert gas will be diluted and blown away by the ventilation streams to the return; Inert gas injection through cut-throughs behind the face, ie at 200 m or beyond behind the face finish line, would result in better goaf inertisation. It can also be observed from the simulations that inertisation even at 500 m behind the face would achieve a better goaf inertisation than that at seals close to the face line.

CFD simulations were also conducted to investigate the effectiveness of goaf inertisation via surface goaf holes in case goaf access via underground roadways becomes difficult or impossible. This is the likely scenario if high level CO were detected in the return and underground workers had to withdraw or due to poor geological conditions/roadway failure, parts of the longwall working become inaccessible.

Figure 9 shows the CFD model layout based on a real case study of an Australian underground coal mine. Due to the collapse of tailgate, the face had to stop and as such signs of spontaneous heating were detected from the goaf. To suppress the development of spontaneous heating in the goaf and thus allowing time for the restoration of the tailgate, inert gas was injected via the vertical boreholes drilled from the surface, as shown in Figure 9.

CFD simulations were carried out to assist the formulation of an optimum strategy for goaf inertisation. Figure 9 also shows the oxygen distribution pattern within the goaf before inert gas inertisation. It can be seen that airflow has penetrated deep into the goaf and a large area can be liable to spontaneous heating.

A range of parametric studies were carried out to investigate the effect of goaf inertisation via the surface goaf holes using nitrogen and boiler gas at different flow rates. Figure 10 shows the goaf oxygen distributions of selected simulations of inertisation strategies. In comparison with the base model, the results indicate that inertisation via surface goaf holes can be very effective in narrowing down the sponcom liable zones. A combination of deep goaf hole (mggh2) with goaf hole (mggh1 or tggh1) can further improve the effectiveness of goaf inertisation, as shown in Figure 10.
10b and c. Studies are continuing in this area to investigate if low inert gas flow rates, i.e, at 0.15 m$^3$/s, can be used to effectively suppress the development of spontaneous heating spots in the goaf.

The improved understanding of the inertisation process has been used in combination with detailed field trials to develop effective proactive inertisation strategies for two Australian underground coalmines, both experienced the threat of goaf heating following the disturbance of geological and operational problems. The implementation of the proactive inertisation strategies tailored specifically to the longwall panel has effectively suppressed the development of spontaneous heatings in the goaf and hence allowed the continuation of coal production.

CONCLUSIONS

In combination with detailed field studies, extensive CFD modelling work has been conducted to investigate the gas flow mechanisms within longwall goafs. These studies have greatly improved the fundamental understanding of goaf gas flow patterns and gas distribution in the longwall goaf and thus help the development of innovative goaf inertisation strategies for both panel seal-off operations and active longwall panels.

The optimum inertisation strategy implemented at the mine was highly successful in converting the goaf environment into an inert atmosphere within a few hours of panel sealing. This represents a major improvement to mine safety compared to typical inertisation practices that were able to achieve goaf inertisation within two to four days after sealing. The study demonstrated that it is feasible to completely inertise the longwall goafs within a few hours of sealing the panel by implementing optimum inertisation strategies.

Investigations have been conducted to develop proactive inertisation strategies to suppress the onset of spontaneous heating in the goaf behind active longwall panels. The studies indicated that inertisation through the cut-through seals at some 200 m behind the face would be more effective than that at close range immediately behind the face line. Goaf inertisation can also be carried out with surface boreholes when underground access becomes prohibitive or impossible. Knowledge obtained from the CFD modelling studies was used in conjunction with field studies to develop proactive goaf inertisation schemes for two Australian coal mines. The implementation of the proactive strategies has been proved highly successful in containing the development of heatings spots in the longwall goafs. Further studies are continuing in a number of areas, including the study of low inert gas flow rates for effective inertisation as well as the use of form injection to reduce the risk of heatings in the active longwall goafs.

The fundamental understanding of inert gas flow patterns and optimum inertisation guidelines developed during the course of the study greatly enhance the safety of coal mines.

ACKNOWLEDGEMENTS

The authors wish to thank ACARP for supporting the study. Thanks are due to the many Australian Coal Mines, management team and individuals for their support and generosity in providing useful information and ground support. The authors are also grateful to their colleagues at CSIRO for advice and assistance during the course of the studies.

REFERENCES


Mining Cadetship

Sustainability through training

February 2005
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Publication date 18 February 2005
Executive Summary

The proposed mining operations management training and education program is embedded within the Australian Qualifications Framework (which brings together schools, colleges and universities). It is a work-site, work-based program supported by training providers and industry mentors. It is a new initiative for the Queensland Mining Industry that seeks to address the forecast deficiencies of appropriately qualified and experienced people in the mining workforce and particularly management, in the future. It is a long-term cultural intervention with the intention of assisting the way in which the industry perceives itself and how the wider community (locally, in Australia and overseas) sees the mining industry.

The initiative will be monitored by a supporting research program to ensure the delivery of the required education and training outcomes. This research will monitor practical and teaching and learning difficulties of delivering the program and assess its community impact inside and outside of the industry. It will also provide a forum for ongoing consideration of curriculum content so that the industry, working collaboratively with technical specialists and training and education providers, can ensure vocational relevance.

This Mining Cadetship program proposal envisages the collaboration of Central Queensland Institute of Technical and Further Education (CQIT), incorporating TAFE Queensland Mining Services (TQMS), the Central Queensland University (CQU) and others as educational providers, with the Queensland Mining Industry Training Advisory Body (QMITAB) and mining companies.

Industry, University and TAFE working together reflects the growing recognition that to best meet the skills development needs of Australia the different sectors must work more closely together. CQU, as a regional university, has a key mission to be actively engaged as a regional resource, providing teaching and research. In discussion with CQIT and others, a networked centre of mining education and training excellence is envisaged with providers working together with industry to best address industry needs.

A networked support base is critical not only to directly supporting cadets but, as importantly, to supporting the teaching staff and mentors (who will play the primary role as educators). Ongoing collaboration with the mining industry is also essential to maintaining the relevance of the cadetship program to the changing needs of the industry. Networked, co-operative engagement across sectors is the key characteristic of the proposed model for development, management, operation and review of the program.

The Mining Cadetship program, commencing with the coal mining industry, provides a platform for the integration of coal and metalliferous mining training and education. It will also help to support the growing links between the mining industries in Australia, China and other countries.
### Glossary

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<tr>
<td>AQF</td>
<td>Australian Qualifications Framework</td>
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<td>AQTF</td>
<td>Australian Qualifications Training Framework</td>
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<td>BMInOpsMgt</td>
<td>Bachelor of Mine Operations Management</td>
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<td>CD ROM</td>
<td>Compact Disc Read Only Memory</td>
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<td>CQIT</td>
<td>Central Queensland Institute of TAFE</td>
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<td>CQU</td>
<td>Central Queensland University</td>
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<td>HECS</td>
<td>Higher Education Contribution Scheme</td>
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<td>ITAB</td>
<td>Industry Training Advisory Body</td>
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<td>QMITAB</td>
<td>Queensland Mining Industry Training Advisory Body</td>
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<td>RCC</td>
<td>Recognition of Current Competency</td>
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<td>RPL</td>
<td>Recognition of Prior Learning</td>
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<td>RTO</td>
<td>Registered Training Organisation</td>
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<td>TAFE</td>
<td>Technical and Further Education</td>
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<td>TQMS</td>
<td>TAFE Queensland Mining Service</td>
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<td>VET</td>
<td>Vocational Education and Training</td>
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1. Introduction

The proposed Mining Cadetship program provides a mining industry integrated vocational education pathway from school into mine management. Distance and flexible education technologies, with Technical and Further Education (TAFE) and university teaching supported by industry mentors, enable mine sites to provide a mini-campus infrastructure. The program provides cadets with the depth and breadth of learning opportunities necessary to enable them to achieve First Class Mine Manager accreditation and a Bachelor degree in Mine Operations Management.

The Mining Cadetship program is primarily targeted at school leavers who want to pursue a career culminating in appointment as a Mine Manager. Nonetheless, the cadetship and its vocational education structure support multiple points of entry to the program and also allow multiple points of exit with an AQF qualification at the level reached.

The cadetship program will be fully industry funded, enabling cadets to complete, as full-time employees and students, the six years of work-based study required. Funding could be provided either through company-sponsored or industry-sponsored cadetships.

The vocational education structure could also be available to those who wish to study whilst employed in the industry as part-time in-service students outside of the cadetship program.

Cadetship programs for careers in mine management are proposed in both underground and open cut mining. It is also anticipated the cadetships would lead to other senior managerial positions in both the coal and metalliferous mining industries – e.g. Technical Services Manager, Training Manager, Safety Manager, Environmental and Occupational Health Manager or Business Services Manager.

The program’s vocational education structure is to be provided through a Centre of Mining Education and Training Excellence which brings together expertise from Central Queensland Institute of TAFE (TAFE Queensland Mining Services) and Central Queensland University (Faculties of Engineering and Physical Systems, Business and Law, Arts, Health and Sciences and Informatics and Communications) and others working in collaboration with the Queensland Mining Industry.

Most importantly it provides multiple points of entry and exit to maximise industry’s return on their investment and training.
2. **Background**

The most important statutory positions required in general coal mining management operations are:

- Mine Manager (First Class) (underground)
- Mine Manager (Second Class – Under Manager) (underground)
- Mine Deputy (underground)
- Open Cut Examiner

These positions require the competencies included in the National Coal Industry Training Package MNC98 and qualifications pursuant to the Australian Qualifications Framework (AQF).

Career paths in the Mining Industry are typically located in regional and remote Australia. It is therefore critical that they are matched with training and educational opportunities that can be directly accessed by the communities of regional and remote Australia. This Mining Cadetship program is structured to achieve this goal.

The Mining Cadetship program concept was developed by a joint Working Group that brought together representatives from the Queensland Mining Industry to work with TAFE and University and other training and education providers. This concept builds on the experience of the highly successful Mining Industry Cadetship Scheme that ran in Queensland throughout the 1980’s. It draws heavily on the work of the Queensland Mining Industry Training Advisory Body (QMITAB) and the Australian Qualification Framework (AQF) that encourages integration and articulation between three education sectors: Secondary Schools, Vocational Education and Training and Higher Education. This ground-breaking work has provided the impetus for the development of a centre of training excellence that brings together TAFE, University and other providers to work with industry.

The Mining Cadetship program is a package that will utilise:

- mine sites as mini-campuses
- industry and training provider learning facilitation and mentoring
- work-based learning
- appropriate financial, pastoral and lifestyle support
- recognised industry and educational qualifications
3. The Australian Qualifications Framework (AQF)

The AQF comprises twelve national qualifications. They are issued in:

- the secondary schools sector
- the vocational education and training sector (Registered Training Providers (RTO) TAFE institutes and private providers)
- the higher education sector (universities)

These qualifications are shown below, grouped according to the educational sector in which they are most commonly issued.

It should be noted that vocational education and training is increasingly provided in the schools sector (VET in Schools). This is recognised at the appropriate Certificate I to IV level and/or as credit towards the Senior Secondary Certificate of Education. In Australia, private and public providers operate across all three sectors.

4. Australian Qualifications Framework by Educational Sector

<table>
<thead>
<tr>
<th>Secondary Schools Sector</th>
<th>Vocational Education and Training Sector</th>
<th>Higher Education Sector</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Doctorate</td>
<td></td>
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<tr>
<td></td>
<td>Masters Degree</td>
<td></td>
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<tr>
<td></td>
<td>Graduate Diploma</td>
<td></td>
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<tr>
<td></td>
<td>Graduate Certificate</td>
<td></td>
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<tr>
<td></td>
<td>Bachelor Degree</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Advanced Diploma / Associate Degree</td>
<td>Advanced Diploma / Associate Degree</td>
</tr>
<tr>
<td></td>
<td>Diploma</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Certificate IV</td>
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<tr>
<td></td>
<td>Certificate III</td>
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<tr>
<td></td>
<td>Certificate II</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Certificate I*</td>
<td></td>
</tr>
</tbody>
</table>

Senior Secondary Certificate of Education

* N.B. Certificate I is not part of the Mining sector qualification structure
5. **Aims of the Mining Cadetship Program**

The aims of the Mining Cadetship Program are to:

- Provide a well supported vocational pathway to a career as a Mine Manager and other mine management positions.
- Contribute to the sustainability of the Mining Industry by supporting relevant skills and knowledge acquisition.
- Provide access to educational, training and employment opportunities to people living in regional and remote locations.
- Support the application of the Australian Qualification Framework to employment in the Mining Industry.

6. **Mining Cadetship Program Structure**

The Mining Cadetship program comprises the following features:

- Six years of predominantly mine-site based study.
- Potential to re-locate to different mine sites in order to support relevant learning.
- On-site mining industry mentoring, and training provider supervision.
- Points of entry:
  a. a primary entry point for post-year 12 secondary school leavers or equivalent
  b. multiple secondary entry points across the full range of the program with certificates and diplomas awarded prior to entry contributing to the final Bachelor degree award.
  c. pre-cadetship studentship
- Selection of candidates as Mining Cadets by a panel of mining industry, training and education members. The panel will consider the following criteria in assessing applications:
  a. Successful completion of a pre-mining cadetship program school studentship, if appropriate
  b. Personal attributes
  c. Career potential in the mining industry
  d. Educational achievements
7. Mining Cadetship Program Models

7.1 Mining Cadetship Program Structure Model - Underground Mining

| Year 6 | Bachelor of Mine Operations Management  
(Minimum 1 year equivalent full-time study)  
Refer to Appendix 12 |
|--------|-----------------------------------------------------------------------------------|
| Year 5 | Advanced Diploma of Underground Coal Mining Management  
Refer to Appendix 10 |
| Year 4 | Diploma of Underground Coal Mining Management  
Refer to Appendix 8 |
| Year 3 | Certificate IV in Underground Coal Mining  
Refer to Appendix 5 |
| Year 2 | Certificate III in Underground Coal Operations  
Refer to Appendix 3 |
| Year 1 | Certificate II in Underground Coal Operations  
Refer to Appendix 1 |
| Pre Cadetship Studentship | Year 11 and/or 12 Secondary School Senior Secondary Certificate of Education &  
Part Certificate II |
## 7.2 Mining Cadetship Program Structure Model - Open Cut Mining

<table>
<thead>
<tr>
<th>Year 6</th>
<th>Bachelor of Mine Operations Management (Minimum 1 year equivalent full-time study) Refer to Appendix 12</th>
</tr>
</thead>
<tbody>
<tr>
<td>Year 5</td>
<td>Advanced Diploma of Surface Coal Mining Management Refer to Appendix 11</td>
</tr>
<tr>
<td>Year 4</td>
<td>Diploma of Surface Coal Mining Management Refer to Appendix 9</td>
</tr>
<tr>
<td>Year 3</td>
<td>Certificate IV in Surface Coal Mining OR Certificate IV in Surface Coal Mining (Open Cut Examiner) Refer to Appendix 6 and 7</td>
</tr>
<tr>
<td>Year 2</td>
<td>Certificate III in Surface Coal Operations Refer to Appendix 4</td>
</tr>
<tr>
<td>Year 1</td>
<td>Certificate II in Surface Coal Operations Refer to Appendix 2</td>
</tr>
<tr>
<td>Pre Cadetship Studentship</td>
<td>Year 11 and/or 12 Secondary School Senior Secondary Certificate of Education &amp; Part Certificate II</td>
</tr>
</tbody>
</table>
8. **AQF Qualification / University-TAFE-School Award / Coal Mining Industry Positions**

<table>
<thead>
<tr>
<th>Bachelor Degree</th>
<th>Mine Manager (First Class)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bachelor of Mine Operations Management</td>
<td>(Underground only)</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Advanced Diploma of Underground Coal Mining Management OR Surface Coal Mining Management</th>
<th>Mine Manager (Second Class)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>(Underground only)</td>
</tr>
<tr>
<td></td>
<td>(Under Manager)</td>
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</tbody>
</table>

<table>
<thead>
<tr>
<th>Diploma of Underground Coal Mining Management OR Surface Coal Mining Management</th>
<th>Mine Deputy – Underground or Open Cut Examiner</th>
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</thead>
</table>

<table>
<thead>
<tr>
<th>Certificate IV in Underground Coal Mining OR Surface Coal Mining (Open Cut Examiner)</th>
<th>Mine Operative</th>
</tr>
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</table>

<table>
<thead>
<tr>
<th>Certificate III in Underground Coal Operations OR Surface Coal Operations</th>
<th>Mine Operative</th>
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</thead>
</table>

<table>
<thead>
<tr>
<th>Certificate II in Underground Coal Operations OR Surface Coal Operations</th>
<th></th>
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</table>

<table>
<thead>
<tr>
<th>Certificate I in Engineering (Pre-vocational)*</th>
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</table>

<table>
<thead>
<tr>
<th>Secondary School Certificate of Education – (Queensland Senior Certificate)</th>
<th></th>
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</thead>
</table>

* not part of the mining qualification structure
9. Training and Education Program – links to University and to Secondary Schools

9.1 The Bachelor of Mine Operations Management (BMinOpsMgt)

Up to the current Advanced Diploma level the curriculum is as determined by the QMITAB. CQIT and CQU will collaboratively deliver the whole program with CQIT taking the primary responsibility from Certificate II up to Diploma level, the two institutions sharing delivery of the Advanced Diploma level and CQU having primary responsibility for the final year, culminating in the award of the Bachelor of Mine Operations Management. It is proposed that the degree could be awarded and the testamur imprinted with CQU, CQIT and QMITAB coats of arms or logos.

The QMITAB, in conjunction with education providers will be responsible for ongoing curriculum development of the combined qualification up to Advanced Diploma level and CQIT, working with the QMITAB, will be responsible for ensuring that cadets meet the appropriate standards, and that these are examined and accredited as required by the mining industry. Recognition of Prior Learning (RPL) or Current Competencies (RCC) will be credited, enabling a CQU Bachelor degree to be awarded with additional study (see appendix 12). This additional study will be assessed by CQU in collaboration with CQIT and the mining industry.

9.2 Recognition of Prior Learning and Recognition of Current Competencies

Through alignment with AQF national standards, the program allows the recognition of cadets’ previous appropriate study and work experience gained through recognised providers and industry up to Advanced Diploma level. This is through formal Recognition of Prior Learning (RPL) or Recognition of Current Competencies (RCC). Cadets seeking credits will be required to demonstrate achievement of each unit’s or course’s learning outcomes.

9.3 Educational Program Format Delivery

Flexible delivery encompassing the internet, CD ROM and other resources will be used as appropriate to course or unit content and required learning outcomes. Courses and units will be supported by face-to-face contact which will include mentors and training program supervisors on mine-sites and off-site at workshops.
9.4 Outline of Bachelor of Mine Operations Management (BMinOpsMgt)

It is proposed that the structure of the post Advanced Diploma element leading to the award of the Bachelor of Mine Operations Management would be broadening knowledge and skills development for cadets with courses determined by the cadet’s employer in negotiation with the cadetship program administrators culminating in the study of an industry based project (refer to Appendix 12). This would allow cadets to holistically demonstrate the necessary knowledge, skills, attributes and attitudes of degree graduates and mining industry professionals.

Programs will also be considered to service other mining vocational pathways e.g. Safety and Environmental Services Management, Training Services Management, Technical Services Management and Business Services Management – in the both the metallicferous and coal mining industry sectors.

The Bachelor of Mine Operations Management would also be available to people wishing to study it who were not part of the Mining Cadetship scheme. It should be noted however that it would require students to be enrolled as full-fee-paying (fee-for-service) students. The program is not intended to be available under the government funded Student Contribution scheme (formerly known as the Higher Education Contribution Scheme - HECS).

9.5 Links to Schools (Studentships)

An important aspect in developing sustainable personnel recruitment and retention is to establish links into local area secondary, and potentially primary schools. This would provide a pathway for students to realise the benefits and opportunities of a career in the mining industry. The availability of a school based studentship that focuses students on the potential for a career in the mining industry in their local area would be an important element in quality personnel recruitment and retention and ultimately industry succession planning. Pre cadetship studentships would enable secondary students with an interest in, and/or expectation of, a career in the mining industry to progress through pre-vocational studies and work experience via a school studentship into the cadetship program.

Successful completion of a studentship would be an important consideration for selection for Mining Cadetships, along with other factors as detailed in Section 6.
A total of twelve (12) units of Competency comprising:

(A) Four (4) mandatory units of competency:
MNCC 1001A  Work Safely
MNCC 1005A  Comply with site work systems / procedures
MNCC 1006A  Conduct local risk assessment
MNCC 1007A  Communicate in the workplace

(B) Eight (8) elective units to be completed:
- a minimum of six (6) units from the Coal Training Package Underground units listed
- a maximum of two (2) units, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages

Underground units
MNCU 1011A  Conduct underground lifting operations
MNCU 1012A  Operate power tram
MNCU 1013A  Conduct rail vehicle operations
MNCU 1014A  Conduct tracked vehicle/plant operations
MNCU 1015A  Conduct wheeled vehicle operations (non-articulated)
MNCU 1016A  Conduct wheeled grader operations
MNCU 1017A  Conduct wheeled vehicle operations (articulated)
MNCU 1037A  Escape from hazardous situation unaided
MNCU 1040A  Install, maintain and recover gas drainage systems
MNCU 1041A  Install, maintain and recover electrical services
MNCU 1042A  Install, maintain and recover water and air systems
MNCU 1049A  Support shotfiring operations
MNCU 1050A  Conduct rotational drilling
MNCU 1053A  Conduct basic strata control operations
MNCU 1060A  Conduct roadway maintenance
MNCU 1061A  Conduct stonedusting operations
MNCU 1062A  Dewater roadways and work areas
MNCU 1063A  Lay and recover rail
MNCU 1064A  Install and maintain explosion barriers
MNCU 1065A  Construct and maintain basic ventilation devices
MNCU 1072A  Conduct feeder breaker operations
MNCU 1073A  Conduct face ventilation operations
MNCU 1077A  Operate longwall ancillary equipment
MNCU 1091A  Maintain lamp cabin operations
MNCU 1101A  Apply spontaneous combustion management measures
General Units

BSBCMN 215A  Participate in environmental work practices
MNC 1004A  Respond to local emergencies and incidents
MNC 1012A  Maintain bathroom hygiene
MNC 1013A  Apply initial response First Aid
MNC 1025A  Access, update and retrieve simple computerised information
MNC 1026A  Operate a computer to produce documents
MNC 1029A  Transfer information through a computer
MNC 1030A  Conduct purchasing
MNC 1031A  Remove, fit and adjust wheel(s)
MNC 1032A  Remove, repair and refit tyres and tubes
MNC 1035A  Apply operational maintenance skills
MNC 1036A  Provide support to electrical tradesperson
MNC 1037A  Service mine plant and equipment
MNC 1038A  Perform basic cutting and welding
MNC 1040A  Operate gantry crane
MNC 1041A  Conduct non-slewing crane operations
MNC 1043A  Conduct dogging operations
MNC 1044A  Conduct basic rigging operations
MNC 1045A  Conduct intermediate rigging operations
MNC 1046A  Conduct basic scaffolding operations
MNC 1047A  Conduct intermediate scaffolding operations
MNC 1048A  Conduct forklift operations
MNC 1049A  Operate elevating work platform
MNC 1050A  Operate vehicle loading crane
MNC 1055A  Extend, retract and maintain conveyor componentry
MNC 1060A  Operate support equipment
MNC 1061A  Operate light vehicle
MNC 1062A  Operate medium vehicle
MNC 1068A  Test operational functions of mine vehicles and equipment
MNC 1070A  Provide deck support for conveyor-car high wall mining operations
MNMUGC 1216A  Conduct skip operations
MNMUGC 1217A  Operate automated winder
MNMUGC 1218A  Operate manual winder
MNMUGC 224A  Conduct Cage operations
MNMUGC 225A  Operate winder for shaft sinking
MNMUGC 226A  Maintain winder equipment
MNMUGC 227A  Inspect and maintain shaft structures
MNMUGC 228A  Monitor, inspect and service ropes and attachments
APPENDIX 2

Program Structure for MNC 20104
Certificate II in Surface Coal Operations

A total of nine (9) units of competency comprising:

(A) Four (4) mandatory units of competency:
- MNCC 1001A  Work Safely
- MNCC 1005A  Comply with site work systems / procedures
- MNCC 1006A  Conduct local risk assessment
- MNCC 1007A  Communicate in the workplace

(B) Five (5) elective units of which:
- a minimum of three (3) units from the specified Open Cut Coal Training Package units listed
- a maximum of two (2) units, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages

- **Open Cut units**
  - MNCO 1013A  Conduct front end loader operations
  - MNCO 1014A  Conduct haul truck operations
  - MNCO 1015A  Conduct bulk water truck operations
  - MNCO 1016A  Conduct grader operations
  - MNCO 1017A  Conduct scraper operations
  - MNCO 1018A  Conduct dozer operations
  - MNCO 1023A  Support bucketwheel system operations
  - MNCO 1024A  Conduct wheeled dozer operations
  - MNCO 1025A  Lay and recover cables and hoses
  - MNCO 1026A  Conduct dewatering operations
  - MNCO 1028A  Conduct conveyor operations
  - MNCO 1029A  Conduct mobile slew conveyor operations
  - MNCO 1032A  Isolate and access plant
  - MNCO 1033A  Conduct conveyor shifting dozer operations
  - MNCO 1036A  Conduct mobile crushing and screening plant operations

- **General Units**
  - BSBCMN 215A  Participate in environmental work practices
  - MNCG 1004A  Respond to local emergencies and incidents
  - MNCG 1005A  Conduct Fire Team Operations
  - MNCG 1010A  Assess and implement life support systems and stabilise casualties
  - MNCG 1011A  Extricate and transport people involved in incidents
  - MNCG 1012A  Maintain bathroom hygiene
  - MNCG 1013A  Apply initial response First Aid
  - MNCG 1025A  Access, update and retrieve simple computerised information
  - MNCG 1026A  Operate a computer to produce documents
  - MNCG 1031A  Remove, fit and adjust wheel(s)
  - MNCG 1032A  Remove, repair and refit tyres and tubes
MNCG 1033A  Conduct lifting operations
MNCG 1035A  Apply operational maintenance skills
MNCG 1036A  Provide support to electrical tradesperson
MNCG 1037A  Service mine plant and equipment
MNCG 1038A  Perform basic cutting and welding
MNCG 1040A  Operate gantry crane
MNCG 1041A  Conduct non-slewing crane operations
MNCG 1042A  Conduct slewing crane operations
MNCG 1043A  Conduct dogging operations
MNCG 1044A  Conduct basic rigging operations
MNCG 1046A  Conduct basic scaffolding operations
MNCG 1048A  Conduct forklift operations
MNCG 1049A  Operate elevating work platform
MNCG 1050A  Operate vehicle loading crane
MNCG 1055A  Extend, retract and maintain conveyor componentry
MNCG 1060A  Operate support equipment
MNCG 1061A  Operate light vehicle
MNCG 1062A  Operate medium vehicle
MNCG 1063A  Operate heavy rigid vehicle
MNCG 1064A  Operate articulated vehicle
MNCG 1068A  Test operational functions of mine vehicles and equipment
MNCG 1070A  Provide deck support for conveyor-car high wall mining operations

➢ Coal Preparation and Treatment

MNCP 1001A  Handle raw coal
MNCP 1002A  Monitor coal preparation plant operations
MNCP 1004A  Treat and dispose of rejects and tailings
MNCP 1005A  Conduct sampling operations
MNCP 1006A  Conduct stockpile dozer operations
MNCP 1007A  Conduct stockpile reclaimer operations
MNCP 1008A  Conduct rail dispatch operations
MNCP 1009A  Perform plant operational maintenance
APPENDIX 3

Program Structure for MNC 30304
Certificate III in Underground Coal Operations

A total of eighteen (18) units of competency made up of:
(A) Twelve (12) units of competency satisfying the criteria for the Certificate II in Underground Coal Operations

(B) Six (6) elective units made up of:
• a minimum of two (2) elective units of competency from the specified Coal Training Package Underground units listed
• a maximum of four (4) units, relevant to the job function, drawn from other endorsed Training Packages.

Note:
• The elective units of competency chosen to satisfy the Certificate III in Underground Coal Operations must be additional to the units of competency achieved to satisfy the Certificate II in Underground Coal Operations.
• The maximum number of units drawn from other endorsed Training Packages must not exceed four (4) units, including units from other endorsed Training Packages included in a Certificate II in Underground Coal Operations.

Underground units
MNCU 1026A Conduct environmental monitoring
MNCU 1037A Escape from hazardous situation unaided
MNCU 1038A Provide aided rescue to endangered personnel
MNCU 1039A Respond to in-seam incident
MNCU 1046A Conduct winding operations
MNCU 1048A Conduct shotfiring
MNCU 1049A Support shotfiring operations
MNCU 1050A Conduct rotational drilling
MNCU 1051A Conduct directional drilling
MNCU 1053A Conduct basic strata control operations
MNCU 1054A Conduct specialised strata control operations
MNCU 1066A Construct and maintain ventilation devices
MNCU 1070A Conduct continuous miner operations
MNCU 1071A Conduct shuttle car operations
MNCU 1073A Conduct face ventilation operations
MNCU 1074A Conduct outburst mining operations
MNCU 1075A Conduct shearer operations
MNCU 1076A Conduct longwall face equipment operations
MNCU 1078A Install and recover longwall equipment
MNCU 1079A Operate breaker line supports
MNCU 1080A Conduct flexible conveyor train (FCT) operations
MNCU 1095A Monitor control processes
MNCU 1101A Apply spontaneous combustion management measures
General Units

MNCG 1001A  Apply the risk management process
MNCG 1005A  Conduct Fire Team Operations
MNCG 1008A  Conduct safety and health investigations
MNCG 1009A  Communicate information
MNCG 1013A  Apply initial response First Aid
MNCG 1027A  Use keyboard skills and advanced functions of software to produce documents
MNCG 1028A  Use advanced functions of software packages to produce documents, reports and worksheet
MNCG 1042A  Conduct slewing crane operations
MNCG 1056A  Install, commission and maintain major conveyor equipment and systems
MNCG 1057A  Repair and splice conveyor belting
MNCG 1069A  Conduct conveyor car high wall mining operations
MNCG 1070A  Provide deck support for conveyor-car high wall mining operations

MNMUGC 1216A  Conduct skip operations
MNMUGC 1217A  Operate automated winder
MNMUGC 1218A  Operate manual winder
MNMUGC 224A  Conduct Cage operations
MNMUGC 225A  Operate winder for shaft sinking
MNMUGC 226A  Maintain winder equipment
MNMUGC 227A  Inspect and maintain shaft structures
MNMUGC 228A  Monitor, inspect and service ropes and attachments
BSBFLM 302A  Support leadership in the workplace
BSBFLM 303A  Contribute to effective workplace relationships
BSBFLM 304A  Participate in work teams
APPENDIX 4

Program Structure for MNC 30104
Certificate III in Surface Coal Operations

A total of fifteen (15) units of competency made up of:
(A) Nine (9) units of competency satisfying the criteria for the Certificate II in Surface Coal Operations

(B) Six (6) elective units made up of:
• a minimum of two (2) elective units of competency from the specified Coal Training Package Open Cut units listed
• a maximum of four (4) units, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages.

Note:
• The elective units of competency chosen to satisfy the Certificate III in Surface Coal Operations must be additional to the units of competency achieved to satisfy the Certificate II in Surface Coal Operations.
• The maximum number of units drawn from other endorsed Training Packages must not exceed four (4) units, including units from other endorsed Training Packages included in a Certificate II in Surface Coal Operations.

➢ Open Cut units
MNCO 1010A  Conduct dragline operations
MNCO 1011A  Conduct burden and coal drilling operations
MNCO 1012A  Conduct rope/shovel operations
MNCO 1013A  Conduct front end loader operations
MNCO 1014A  Conduct haul truck operations
MNCO 1015A  Conduct bulk water truck operations
MNCO 1016A  Conduct grader operations
MNCO 1017A  Conduct scraper operations
MNCO 1018A  Conduct dozer operations
MNCO 1019A  Conduct surface miner operations
MNCO 1020A  Conduct auger miner operations
MNCO 1021A  Conduct bucketwheel operations
MNCO 1022A  Conduct spreader operations
MNCO 1024A  Conduct wheeled dozer operations
MNCO 1027A  Conduct hydraulic shovel operations
MNCO 1030A  Conduct control centre operations
MNCO 1033A  Conduct conveyor shifting dozer operations
MNCO 1040A  Conduct shotfiring operations
MNCO 1041A  Support shotfiring operations
General Units
BSBCMN 313A  Maintain environmental procedures
BSBFLM 302A  Support leadership in the workplace
BSBFLM 303A  Contribute to effective workplace relationships
BSBFLM 304A  Participate in work teams
MNCG 1001A  Apply risk management processes
MNCG 1005A  Conduct Fire Team Operations
MNCG 1008A  Conduct safety and health investigations
MNCG 1009A  Communicate information
MNCG 1010A  Assess and implement life support systems and stabilise casualties
MNCG 1011A  Extricate and transport people involved in incidents
MNCG 1013A  Apply initial response First Aid
MNCG 1027A  Use keyboard skills and advanced functions of software packages to produce complex documents
MNCG 1029A  Transfer information through a computer
MNCG 1030A  Conduct purchasing
MNCG 1045A  Conduct intermediate rigging operations
MNCG 1047A  Conduct intermediate scaffolding operations
MNCG 1056A  Install, commission and maintain major conveyor equipment and systems
MNCG 1057A  Repair and splice conveyor belting
MNCG 1065A  Operate multi combination vehicles on mine sites
MNCG 1068A  Test operational functions of mine vehicles and equipment
MNCG 1069A  Conduct conveyor-car high wall mining operations
MNCG 1070A  Provide deck support for conveyor-car high wall mining operations

Coal Preparation and Treatment
MNCP 1003A  Control coal preparation plant operations
MNCP 1005A  Conduct sampling operations
MNCP 1007A  Conduct stockpile reclaimer operations
APPENDIX 5

Program Structure for MNC 40304
Certificate IV in Underground Coal Mining

Successful completion of eleven (11) units of competency made up of:

(A) Eight (8) mandatory units of competency:
- MNCG 1002A  Implement and apply the risk management processes
- MNCU 1104A  Apply the spontaneous combustion management plan
- MNCU 1108A  Apply and monitor the ventilation management plan
- MNCU 1113A  Apply and monitor the gas management plan
- MNCU 1133A  Apply and monitor the strata management plan
- MNCU 1138A  Apply and monitor mine transport systems and production equipment
- MNCU 1143A  Apply and monitor mine services and infrastructure systems
- MNCU 1153A  Apply and monitor mine emergency preparedness and response systems

(B) Three (3) elective units from the Technical Management and General Management units listed:

- **Technical Management**
  - MNCU 1037A  Escape from hazardous situation unaided
  - MNCU 1038A  Provide aided rescue to endangered personnel
  - MNCU 1039A  Respond to in-seam incident
  - MNCU 1044A  Conduct special roadway operations
  - MNCU 1045A  Recover equipment
  - MNCU 1048A  Conduct shotfiring
  - MNCU 1049A  Support shotfiring operations
  - MNCU 1070A  Conduct continuous miner operations
  - MNCU 1071A  Conduct shuttle car operations
  - MNCU 1074A  Conduct outburst mining operations
  - MNCU 1075A  Conduct shearer operations
  - MNCU 1076A  Conduct longwall face equipment operations
  - MNCU 1118A  Apply and monitor the gas drainage management plan
  - MNCU 1123A  Apply and monitor the outburst management plan
  - MNCU 1128A  Apply and monitor the inrush management plan

- **General Management**
  - MNCG 1007A  Implement and monitor health and hygiene management systems
  - MNCG 1008A  Conduct safety and health investigations
  - MNCG 1009A  Communicate information
  - MNCG 1028A  Use advanced functions of software packages to produce documents, reports and worksheets
  - BSBCMNN 402A  Develop work priorities
  - BSBCMNN 404A  Develop teams and individuals
  - BSBCMNN 410A  Coordinate implementation of customer service strategies
  - BSBCMNN 412A  Promote innovation and change
  - BSBFLM 402A  Show leadership in the workplace
BSBFLM 403A  Manage effective workplace relationships
BSBFLM 404A  Lead work teams
BSBFLM 405A  Implement operational plan
BSBFLM 409A  Implement continuous improvement
APPENDIX 6

Program Structure for MNC 40204
Certificate IV in Surface Coal Mining

Successful completion of fourteen (14) units of competency made up of:

(A) Six (6) mandatory units of competency
- MNCG 1002A Implement and apply the risk management processes
- MNCG 1009A Communicate information
- MNCG 1105A Apply the mine occupational health and safety management plan
- MNCO 1103A Apply pit plan
- MNCO 1115A Apply and monitor surface mine emergency preparedness and response procedures
- MNCO 1045A Apply and monitor environmental management policies, plans and procedures

(B) Eight (8) elective units including:
- a minimum of two (2) units from the specified Technical Management units
- a minimum of four (4) units from the specified General Management units
- a maximum of two (2) units, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages.

➤ Technical Management
- MNCG 1005A Conduct Fire Team Operations
- MNCG 1007A Implement and monitor health and hygiene management systems
- MNCG 1008A Conduct safety and health investigations
- MNCO 1031A Coordinate conveyor system shift
- MNCO 1040A Conduct shotfiring operations
- MNCO 1041A Support shotfiring operations
- MNCO 1043A Monitor the interaction of heavy and light vehicles and mining equipment
- MNCO 1046A Apply and monitor systems and methods of mining
- MNCO 1047A Manage the interaction of light and heavy vehicles and mining equipment
- MNCO 1107A Apply and monitor the site water management plan
- MNCO 1112A Apply and monitor the site stockpile management plan
- MNCO 1117A Apply and monitor site plant and resource management plan
- MNCO 1118A Supervise coal processing operations
- MNCO 1122A Apply and monitor site waste and by-products management plan
- MNCO 1127A Apply and monitor site plant, equipment and infrastructure maintenance management plan

➤ General Management
- MNCG 1028A Use advanced functions of software packages to produce documents, reports and worksheets
- MNCG 1113A Apply quality management system
- BSBCMN 402A Develop work priorities
BSBCMN 404A  Develop teams and individuals
BSBCMN 410A  Coordinate implementation of customer service strategies
BSBCMN 412A  Promote innovation and change
BSBFLM 402A  Show leadership in the workplace
BSBFLM 403A  Manage effective workplace relationships
BSBFLM 404A  Lead work teams
BSBFLM 405A  Implement operational plan
BSBFLM 409A  Implement continuous improvement
APPENDIX 7

Program Structure for MNC 40104
Certificate IV in Surface Coal Mining (Open Cut Examiner)

Successful completion of eleven (11) units of competency made up of:

(A) Eight (8) mandatory units of competency (management and technical)

- MNCG 1002A Implement and apply the risk management processes
- MNCG 1007A Implement and monitor health and hygiene management systems
- MNCG 1008A Conduct safety and health investigations
- MNCO 1115A Apply and monitor surface mine emergency preparedness and response procedures
- MNCO 1040A Conduct shotfiring operations (mandatory in NSW)
- OR
- MNCO 1041A Support shotfiring operations (mandatory in Queensland)
- MNCO 1042A Examine and maintain mine safety
- MNCO 1043A Monitor the interaction of heavy and light vehicles and mining equipment
- MNCO 1046A Apply and monitor systems and methods of mining

(B) Three (3) elective units including:

- a minimum of one (1) general unit
- a minimum of one (1) unit drawn from the coal open cut units of competency, and
- one (1) unit, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages.

- General

  - MNCG 1009A Communicate information
  - MNCG 1025A Access, update and retrieve simple computerised information
  - MNCG 1026A Operate computer to produce documents
  - MNCG 1027A Use keyboard skills and advanced functions of software packages to produce complex documents
  - MNCG 1028A Use advanced functions of software packages to produce documents, report and worksheets
  - MNCG 1029A Transfer information through computer
  - BSBCM 402A Develop work priorities
  - BSBCM 404A Develop teams and individuals
  - BSBCM 410A Coordinate implementation of customer service strategies
  - BSBCM 412A Promote innovation and change
  - BSBFLM 402A Show leadership in the workplace
  - BSBFLM 403A Manage effective workplace relationships
  - BSBFLM 404A Lead work teams
  - BSBFLM 405A Implement operational plan
  - BSBFLM 409A Implement continuous improvement
## Open Cut

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<td>MNCO 1010A</td>
<td>Conduct dragline operations</td>
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<td>MNCO 1011A</td>
<td>Conduct burden and coal drilling operations</td>
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<tr>
<td>MNCO 1012A</td>
<td>Conduct rope/shovel operations</td>
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<td>MNCO 1013A</td>
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<td>MNCO 1016A</td>
<td>Conduct grader operations</td>
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<td>MNCO 1017A</td>
<td>Conduct scraper operations</td>
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<td>MNCO 1018A</td>
<td>Conduct dozer operations</td>
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<tr>
<td>MNCO 1045A</td>
<td>Apply and monitor environmental management policies, plans</td>
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<tr>
<td></td>
<td>and procedures</td>
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</table>
APPENDIX 8

Program Structure for MNC 50204
Diploma of Underground Coal Mining Management

Successful completion of eleven (11) units of competency made up of:

(A) Eight (8) mandatory units of competency

- MNCG 1002A Implement and apply the risk management processes
- MNCU 1103A Implement the spontaneous combustion management plan
- MNCU 1107A Implement the ventilation management plan
- MNCU 1112A Implement the gas management plan
- MNCU 1132A Implement strata management plan
- MNCU 1137A Implement mine transport systems and production equipment
- MNCU 1142A Implement mine services and infrastructure systems
- MNCU 1152A Implement mine emergency management system

(B) Three (3) elective units from General and/or Technical Management units listed:

- **Technical Management**
  - MNCG 1007A Implement and monitor health and hygiene management systems
  - MNCG 1081A Implement, monitor, rectify and report on contracts
  - MNCG 1082A Implement, monitor, rectify and report on inventory control system
  - MNCU 1048A Conduct shotfiring
  - OR
  - MNCU 1049A Support shotfiring operations
  - MNCU 1109A Manage, operate and maintain the mine ventilation system
  - MNCU 1117A Implement the gas drainage management plan
  - MNCU 1122A Implement the outburst management plan
  - MNCU 1127A Implement the inrush management plan

- **General Management**
  - MNCG 1008A Conduct safety and health investigations
  - MNCG 1009A Communicate information
  - BSBFLM 501A Manage personal work priorities and professional development
  - BSBFLM 502A Provide leadership in the workplace
  - BSBFLM 503A Establish effective workplace relationships
  - BSBFLM 504A Facilitate work teams
  - BSBFLM 505A Manage operational plan
  - BSBFLM 509A Promote continuous improvement
  - BSBFLM 510A Facilitate and capitalise on change and innovation
  - BSBFLM 511A Develop a workplace learning environment
  - BSBSBM 401A Establish business and legal requirements
  - BSBSBM 402A Undertake financial planning
  - BSBSBM 406A Manage finances
Successful completion of fourteen (14) units of competency made up of:

(A) Two (2) mandatory units of competency
- MNCG 1002A  Implement and apply the risk management processes
- MNCG 1103A  Implement and maintain management systems to control risk

(B) Twelve (12) elective units to be completed including:
- a minimum of five (5) units from the specified Technical Management units
- a minimum of four (4) units drawn from the specified General Management units
- a maximum of three (3) units, relevant to the job function, drawn from the Coal Training Package or other endorsed Training Packages.

➢ Technical Management
- MNCG 1082A  Implement, monitor, rectify and report on inventory control system
- MNCO 1047A  Manage the interaction of light and heavy vehicles and mining equipment
- MNCO 1044A  Manage laser levelling of operating plant
- MNCO 1046A  Apply and monitor systems and methods of mining
- MNCO 1102A  Implement pit plan
- MNCO 1106A  Implement the site water management plan
- MNCO 1111A  Implement the stockpile management plan
- MNCO 1116A  Implement mine plant and resource management plan
- MNCO 1121A  Implement site waste and by-product management plan
- MNCO 1126A  Implement and maintain the site plant, equipment and infrastructure maintenance plan
- MNCO 1131A  Implement mine services systems
- MNCO 1142A  Implement mine fixed plant and infrastructure systems

➢ General Management
- MNCG 1007A  Implement and monitor health and hygiene management systems
- MNCG 1081A  Implement, monitor, rectify and report on contracts
- MNCG 1119A  Manage major incidents and emergencies
- BSBFLM 501A  Manage personal work priorities and professional development
- BSBFLM 502A  Provide leadership in the workplace
- BSBFLM 503A  Establish effective workplace relationships
- BSBFLM 504A  Facilitate work teams
- BSBFLM 505A  Manage operational plan
- BSBFLM 509A  Promote continuous improvement
- BSBFLM 510A  Facilitate and capitalise on change and innovation
- BSBFLM 511A  Develop a workplace learning environment
- BSBSBM 401A  Establish business and legal requirements
- BSBSBM 402A  Undertake financial planning
- BSBSBM 406A  Manage finances
APPENDIX 10

Program Structure for MNC 60204
Advanced Diploma of Underground Coal Mining Management

Successful completion of eleven (11) units of competency made up of:

(A) Eight (8) mandatory units of competency
- MNCG 1003A Establish the risk management system
- MNCU 1102A Establish the spontaneous combustion management plan
- MNCU 1106A Establish the ventilation management plan
- MNCU 1111A Establish the gas management system
- MNCU 1131A Establish the mining method and strata management systems
- MNCU 1136A Establish mine transport systems and production equipment
- MNCU 1141A Establish mine services and infrastructure systems
- MNCU 1151A Establish mine emergency preparedness and response systems

(B) Three (3) elective units from the units listed:

- General Management
  - MNCU 1109A Manage, operate and maintain the mine ventilation system
  - MNCU 1116A Establish the gas drainage management plan
  - MNCU 1121A Establish the outburst management plan
  - MNCU 1126A Establish the inrush management plan
  - MNCG 1006A Incorporate health and hygiene factors into mine management
  - MNCG 1101A Establish and maintain the environmental management system
  - MNCG 1102A Establish the statutory compliance management system
  - MNCG 1107A Establish and maintain the mine Occupational Health and Safety Management system
  - MNCG 1125A Initiate, monitor and supervise contracts
  - MNCG 1126A Conduct business negotiations
  - BSBMGT 503A Prepare budgets and financial plans
  - BSBMGT 601A Contribute to strategic direction
  - BSBMGT 603A Review and develop business plans
  - BSBMGT 605A Provide leadership across the organisation
  - BSBMGT 607A Manage knowledge and information
  - BSBMGT 608A Manage innovation and continuous improvement
APPENDIX 11

Program Structure for MNC 60104
Advanced Diploma of Surface Coal Mining Management

Successful completion of fourteen (14) units of competency made up of:

(A) Six (6) mandatory units of competency
- MNCG 1003A  Establish the risk management system
- MNCG 1101A  Establish and maintain the environmental management system
- MNCG 1102A  Establish the statutory compliance management system
- MNCG 1107A  Establish and maintain the mine Occupational Health and Safety management system
- MNCG 1111A  Establish and maintain the quality system
- MNCG 1116A  Establish mine emergency preparedness and response systems

(B) Eight (8) elective units including:
- four (4) units from the specified Technical Management units
- four (4) units from the specified General Management units

- Technical Management
  - MNCO 1101A  Plan pit development
  - MNCO 1105A  Establish the mine water management system
  - MNCO 1110A  Establish the mine stockpile management system
  - MNCO 1120A  Establish waste and by-product management system
  - MNCO 1125A  Establish plant, equipment and infrastructure maintenance system
  - MNCO 1130A  Establish mine services system
  - MNCO 1135A  Establish ground control and slope stability systems
  - MNCO 1140A  Establish the mine infrastructure and fixed plant systems

- General Management
  - MNCG 1119A  Manage major incidents and emergencies
  - MNCG 1125A  Initiate, monitor and supervise contracts
  - MNCG 1126A  Conduct business negotiations
  - BSBMGT 601A  Contribute to strategic direction
  - BSBMGT 603A  Review and develop business plans
  - BSBMGT 605A  Provide leadership across the organisation
  - BSBMGT 607A  Manage knowledge and information
  - BSBMGT 608A  Manage innovation and continuous improvement
  - BSBMGT 503A  Prepare budgets and financial plans
APPENDIX 12

Program Structure/Content – Bachelor of Mine Operations Management

The content of this final qualification component will be determined by the cadet’s employer in negotiation with the Industry Sponsor / Industry Mentor and the Academic Mentor. The generic structure is an industry based Project (comprising a two part project) plus from six (6) to ten (10) courses from specialty packages: Engineering, Sustainable Development, Environmental Science, Business, Law, Safety, and Information Technology. These courses could be drawn from but will not be restricted to, the following selection:

### Engineering

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<td>ENTA 12011</td>
<td>CAD &amp; Design</td>
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<td>ENTA 12013</td>
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<td>Materials in Service</td>
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<td>ENPO 20004</td>
<td>Process Simulation, Modelling &amp; Optimisation</td>
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Sustainable Development
EVST 20009  Thinking Systemically and Sustainably
MGMT 20083  Models of Sustainable Development
GEOG 11024  Conservation in Australia

Environmental Science
ENEV 20001  Air Quality Management 1
ENEV 20002  Waste Minimisation and Cleaner Production
ENEV 20005  Environmental Planning and Economics
ENEV 20007  Energy, Resources and Environment
ENEV 20008  Environmental Engineering
ENVR 11011  Environmental Science A
ENVR 11012  Environmental Science B
ENVR 20001  Science and the Environment
ENVR 19007  Environmental and Social Impact Assessment
EVST 19008  Environmental Management
EVST 19020  Environmental Management Systems
EVST 20002  Land Management and Rehabilitation (This course has a residential school component)
EVST 20003  Environmental Risk Management (This course has a residential school component)
EVST 20012  Water Management 1 (This course has a residential school component)
EVST 20021  Environmental Management Systems and Sustainable Development

Business
MGMT 19103  Logistics Management
MGMT 19104  Cont Purchasing and Mats Mgt
MGMT 19105  Quality Management
MGMT 19114  Strategic Management
ACCT 11059  Using Acc. for Decision Making
ECON 19032  Microeconomics
ECON 19033  Macroeconomics
HRMT 11011  Human Resources in Orgs
LAWS 19031  Mercantile Law
HRMT 19020  Managing Organisational Change
HRMT 19021  Australian Industrial Relations
MGMT 20124  People, Work and Organisations
MGMT 20121  Leaders and Organisations
MGMT 20094  Organisational Analysis
MGMT 20093  Managing in a Dynamic Environment
MGMT 20085  Managing Operational Effectiveness
MGMT 12097  Industrial Relations
HRMT1 20007  Human Resource Management
ACCT 19083  Corporate Governance and Ethics
COMM 12017  Public Relations Management
COMM 12025  Organisational Communication
MGMT 20112  Strategy Development and Initiatives

**Law**

LAWS 20039  Environmental Law and Regulation
LAWS 20038  International Business Law
LAWS 20037  Law for Management
LAWS 19037  Law for Global Business
LAWS 11046  Law and the Environment

**Safety**

OCHS 12005  Risk Mgt & Safety Technology
OCHS 12015  Law & Mgt of OHS
OCHS 13008  Human Factors
OCHS 13010  Applied Worksite Analysis (Has a block learning component in Rockhampton)
OCHS 13011  Occupational Rehab & Compensation
OCHS 13016  Occupational Health, Hygiene & Toxicology

**Information Technology**

COIS 11014  Engineering Computing
COIS 20024  Systems Management Overview (Requires Internet access)
COMM 12020  Communication in the Digital Age
# APPENDIX 13

## Web Links

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<thead>
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<td>Anglo Coal</td>
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<td>Australian Coal Association</td>
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<td><a href="http://www.macarthurcoal.com.au">http://www.macarthurcoal.com.au</a></td>
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Moorvale Coal  http://www.macarthurcoal.com.au
New Acland Coal  http://www.newhopecoal.com.au
New Hope Corporation Ltd  http://www.newhopecoal.com.au
New Oakleigh Collieries Pty Ltd  http://www.newhopecoal.com.au
Norwich Park Mine  http://www.bmacoal.com
Oaky Creek Coal Pty Ltd  http://www.mim.co.au/coal.html
Pacific Coal  http://www.pacificcoal.com
Peak Downs Mine  http://www.bmacoal.com
Queensland Resource Council  http://www.qrc.org.au
Rio Tinto Coal Australia  http://www.pacificcoal.com.au
Roche Mining  http://www.intergen.com/australia.html
Saraji Mine  http://www.bmacoal.com
TAFE Queensland Mining Service (TQMS)  http://www.cqit.net/mining
Tarong Coal Pty Ltd  http://www.pacificcoal.com.au
The Moura Joint Venture  http://www.anglocoal.com.au
Thiess Mining  http://www.thiess.com.au
Xstrata Coal Queensland Pty Ltd  http://www.mim.co.au/coal.html